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## FOREWORD

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## RESEARCH AND DEVELOPMENT IN SOVIET NONFERROUS METALLURGY

[The following are translations of portions of the Russian book Metallurgiya SSSR, 1917-1957, (USSR Metallurgy, 1917-1957), edited by Academician I. P. Bardin, and published by the State Scientific and Technical Publishing House for Literature on Ferrous and Nonferrous Metallurgy, Moscow, 1958.]

### NONFERROUS METALLURGY DURING THE SOVIET REGIME

#### THE RESTORATION PERIOD

Pages 399-413

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The nonferrous metallurgy of pre-Revolutionary Russia was one of the most backward branches of industry. The Great October Socialistic Revolution established the conditions for rapid development of nonferrous metallurgy.

All enterprises of nonferrous metallurgy were nationalized in the period from May to October 1918. Restoration of production was started.

The restoration of nonferrous metallurgy which had been started was interrupted by civil war.

The nonferrous mining industry, which was located at that time chiefly in the Urals, in Siberia, and the Transcaucasus, was almost entirely in the hands of counterrevolutionary forces and interventionists and was destroyed by them to a considerable extent.

There remained at the disposal of Soviet authority a few plants for processing nonferrous metals in Leningrad, Kol'chugina (Vladimirskaya Oblast') and Tula; they too were in a state of decay.

There are no accurate figures on the state of nonferrous metallurgy in this period. According to the approximate figures of Professor V. Ye. Potresov [1], the production of copper in the territory of the RSFSR [Russian Soviet Federated Socialistic Republic] amounted to: in 1918-2,053 tons, in 1919-200 tons, and in 1920-about 2000 tons, while lead [production] was respectively 89, 200, and 300 tons.

The level of production of heavy industry in 1917-1920 changed as follows:

Year	Level of Production of Heavy Industry, Per Cent
1913	100.0
1917	62.7
1918	34.5
1919	25.8
1920	13.8

At the end of 1919 there was a radical change in the course of the civil war.

The Ninth Party Congress recognized the necessity of the compilation and unswerving implementation of a unified economic plan for the nation.

On the basis of the decisions of the VIII All-Russian Congress of Soviets, the State Plan for the Electrification of Russia, the plan for the restoration and the development of the national economy, the plan for the GOELRO [Gosudarstvennaya komissiya po elektrofikatsii Rissii - State Commission for the Electrification of Russia].

According to this plan, which was calculated for a lengthy period (10-15 years), the smelting of copper was to reach 80,000 tons and aluminum 10,000 tons. A large increase was also stipulated in the production of lead and zinc.

In carrying out the plan of the GOELRO, the first item was the restoration of the Kalatinskiy Copper Plant in the Urals which began regular production of copper on 5 May 1922. The restoration of this plant, like that of the majority of the other mining and metallurgical enterprises of nonferrous metallurgy, proceeded under exceedingly difficult conditions. Likewise, there was no market for copper, since the military stocks of copper covered the small demand for it. Under conditions of an acute crisis in production "nobody would give a pood of grain for a pood of copper" [Translator's note: A pood was equivalent to 16.38 kilograms, or 36 pounds]. In spite of difficult conditions, the plant was restored and successfully developed. For the next few years (1923-1924) this plant was almost the only supplier of copper in the nation. At the same time, the extration of ore in the Northern Caucasus, in the Sadon Mines, and the smelting of lead from it in the Alagir Lead Plant were restored.

In the Urals, the small Pyshma-Klyuchevskiy Plant of the Karabash Copper Smelter, one of the largest plants in the Urals, was restored and put into production on 28 July 1925. The Tanalyk-Baymak Copper Plant in Bashkiria was reopened for work in March 1925. A shaft furnace was built and lead was smelted on a small scale in the Ekibastuz Lead Plant whose construction had not been completed by concessionaires. (See Note). From ores and partial concentrates shipped in from Ridder). The restoration work was accompanied by reconstruction. Thus, for example, the processing of gold-bearing ores was mastered at



the Ridder Ore Concentration Plant. An experimental electrolytic plant was also built for producing zinc.

As a result of the restoration of these plants, the extraction of ore and the production of heavy nonferrous metals was increased markedly. Thus, the smelting of crude [Translator's note: Presumably matte copper] copper in 1924/1925 amounted already to 9,024 tons compared with 1,758 tons in 1922/1923; lead--1,021 tons compared with 307 tons; and zinc 1,492 tons compared with 187 tons. However, the absolute levels of production achieved in these metals was still very low and amounted to a total of 30 per cent of the pre-Revolutionary level (1913) for copper, 40 per cent for gold, 50 per cent for zinc, and only the level of production of lead, which had been negligible up to the Revolution, reached 87 per cent. (/See note/ The figures for zinc for 1913 were given only for the North Caucasus Alagir Plant, excluding Poland.)

The growing demand for nonferrous metals was covered essentially by imports from capitalistic nations.

The turning point to the development of the restorative process is nonferrous metallurgy took place after the XIV Party Congress (December 1925) -- the Congress for industrializing the nation.

The XIV Party Congress devoted particular attention to the development of nonferrous metallurgy. Electrification of the nation, the development of machine building, aviation manufacturing, chemicals, and other branches of heavy industry could not be carried out without significant increases in the production of nonferrous metals.

After the XIV Congress, the Government undertook a series of measures to develop non-ferrous metallurgy--a special interdepartmental commission, the KOMTSVETMETFOND [Abbreviation not known, apparently the Commission on Nonferrous Metals Resources] was established under the Presidium of the VSNKh [Vyschiy Sovet Narodnogo Khozyaystva Supreme Council of the National Economy], and a special fund was set up to finance it.

In 1925 a resolution was adopted concerning the renewal of work in a number of copper plants in the Urals and the construction of the Krasnoural'sk Plant in the Urals, and the Atbasar and Ridder Plants in Kazakhstan. In accordance with this decree [POSTANOVLENIE] the construction of a new lead plant in the Altay (Ridder) was started, the Karsakpay Copper Smelter was completed on the base of rich copper ore from Dzhezkazgan and the construction of the new Krasnoural'sk Copper Plant was started in the Urals. All this led to a marked growth in the extraction of ore and the production of nonferrous metals. By the beginning of the FIRST FIVE-YEAR PERIOD, nonferrous metallurgy had approached the pre-revolutionary level of 1913 even though it had not reached it.

Thus, in 1926-1927, the production of crude copper amounted to 29,200 tons as compared with 31,100 tons in 1913, zinc--2,700 tons against 2,900 tons, and lead--1,200 tons against 1,500 tons. (/see note/ Within the boundaries of the USSR (33,700 tons of copper were produced in the whole of Russia).

The restoration period in nonferrous metallurgy required considerably more time than it did in a number of other branches of industry and in industry as a whole.

The basic reason for this are the following:

1. A sharp lag in the raw materials base which was at an even lower level, in respect to the absolute size of reserves, than in 1913 when the reserves of copper in the deposits were estimated at 627,000 tons of copper, 500,000 tons of lead, and 1,100,000 tons of zinc [2].
2. A lack of skilled workers and engineering-technical personnel.
3. Difficulties encountered in restoring enterprises in remote, uninhabited regions.

The levels of production achieved did not meet the demands of the nation, which were covered by imports.

#### THE FIRST, SECOND, AND THIRD FIVE-YEAR PLANS

With the transition to the five-year plans, the history of the development of nonferrous metallurgy can be arbitrarily divided into three basic periods: the period of the prewar five-year plans, the years of World War II, and the postwar years.

The First Five-Year Plan was characterized by forced development of heavy industry, in the first instance, machine building. The large growth in these branches of industry, in particular machine building, made great demands on nonferrous metallurgy. Therefore, extensive construction of new mines and plants was started along with the reconstruction of operating enterprises during the first Five-Year Plan.

A total of about 1.5 billions of rubles was invested in nonferrous metallurgy in the First Five-Year Plan. Investments in the copper and the lead-zinc branches of nonferrous metallurgy during this period were 7.4 times the total of capital investments in those branches as of 1 October 1928. In particular, large sums were also directed into geological prospecting projects.

In the period from 1928-1932 extensive prospecting projects in the search for copper were carried out in Kazakhstan (Kounrad, Dzhezkazgan, Altay), and in the Urals. The search for lead and zinc was carried out in Kazakhstan (in the Kara Tau Mountain Range, and the mining areas of the Altay).

Raw material for aluminum was developed in the Tkhevinsk Region (in the northwestern part of the European USSR).

As a result of the geological prospecting work new large reserves of copper, lead, and zinc were found. The reserves of copper (in categories A + B + C) had increased in 1932 as compared with 1929 by 3.75 times, the reserves of lead by 1.5 times and the reserves of zinc by 2.7 times [2].

During the First Five-Year Period the following were completed and put into operation: the Krasnoural'sk Copper Smelter (1931), the zinc plants which use retorts for recovering the metal from vapor at Konstantinovka in the Ukraine (1930), at Belovo in Western Siberia

(1931), the Skhote-Alin' Lead Plant in the Far East (1932) and the Volkhov Aluminum Plant (1932).

In the period 1929-1932 copper ore concentration plants were put into operation in the Urals (Korovgrad, Pyshma, Krasnoural'sk) and in Kazakhstan (Karsakpay).

Construction was begun on the Chimkent Lead Plant and the Bolkhash Copper Smelter Plant in Kazakhstan, the Chelyabinsk Zinc Plant, the Dneprovsk Aluminum Plant, the Ufeley Nickel Plant, and others.

During these years the technological level of nonferrous metallurgy also rose markedly. Operating enterprises were reconstructed to a large extent. Manual drilling was largely replaced by mechanized drilling. Ore lifting was mechanized in almost all shafts. Dressing of nonferrous metal ores was widely introduced, which permitted the exploitation of lean ores in copper and lead-zinc deposits. Reverberative smelting was introduced in copper metallurgy in the Krasnoural'sk and partially in the Kalatinskiy (Korovgrad) and the Karabash Plants. Establishing an industry in the rare metals and hard alloys was started in the First Five-Year Plan. The production of aluminum from bauxite was mastered in 1932.

During the First five-Year Plan the production of all the basic nonferrous metals increased by several times. The production of copper in 1932 (45,000 tons) was 2.4 times greater than in 1927-1928, and 1.5 times the 1913 level. The production of lead rose respectively by 8.1 and 15 times, and zinc by 6.2 and 4.4 times.

By the end of the First-Five-Year Plan Period the absolute level of copper production of copper, lead, and zinc in the USSR was a fraction [literally several times lower] than in the United States and in other capitalist countries.

The following factors exerted a negative effect on the development of nonferrous metallurgy during these years.

The first factor was the noncompletion of the restoration process, which was continued up to 1929-1931. Thus, for example, one of the largest Altay mines, at Zyryanovsk, was flooded and ruined by concessionaires, was pumped out and restored at the end of 1931, and was put into operation only in February 1932.

The second factor was the sharp nonfulfillment of the scheduled dates for new construction and the commissioning of new projects as stipulated by the plan [3].

The lag in capital construction is explained by the scattering of capital investments among several projects [literally objects], the discrepancy between the material resources, (particularly an acute shortage of imported equipment) available and the amount of capital construction specified by the plan.

The third factor was imperfect technology in the processing of ores and the concentration of nonferrous metals, as a result of which the losses in metal during processing of ores amounted to 40-45 per cent for copper, 45-50 per cent for zinc, and more than 50 per cent for lead.

Finally, there was a serious shortage of workers, who were poorly skilled and had a high turnover. Thus, for example, at Ridder in 1931-1932, the personnel on hand changed four times a year. The level of production of nonferrous metals achieved in 1932 was far from covering the growing demands for them. The relationship between production and the actual consumption of the basic nonferrous metals (in thousands of tons) at the beginning and the end of the First Five-year Plan are presented in Table 1.

TABLE 1

Metals	1928 - 1929		1932	
	Consumption	Production	Consumption	Production
Copper	55.0	35.5	69.9	45.0
Lead	50.0	5.5	47.1	18.7
Zinc	37.0	3.0	31.0	13.7
Aluminum	-----	-----	17.7	0.9

Thus, in 1932, the covering of consumption through imports amounted to: about 36 per cent for copper, about 56 per cent for zinc, and 60 per cent for lead.

In the Second Five-Year Plan, which had the established task of completing the technological reconstruction of the national economy of the USSR and the extensive mastery of new technology and new production it was essential to achieve a more rapid tempo in the further growth of production of nonferrous metals. The consumption of nonferrous metals in 1937 as compared with 1932 was to increase: 3.65 times for copper, 3.77 times for lead, 4 times for zinc, and 2.41 times for aluminum [1].

In accordance with the resolutions of the SVII Congress of the VKP(b) [Vsesoyuznaya Kommunisticheskaya Partiya (bol'shevikov) - All-Union Communist Party (Bolsheviks)] in respect to the Second Five-Year Plan, the tempo of growth in the production of copper, lead, and zinc was set higher than for other branches of the national economy. Thus, with a general growth of industrial production of 2.1 times, cast iron of 2.6 times, steel of 2.9 times, the production of copper was to increase by 3 times, lead by 6.2 times, and zinc by 6.5 times.

In order to ensure such a large growth of the production of nonferrous metals, it was essential to expand the raw materials bases materially. To obtain a preliminary preparation for an [expansion] in minerals, the tempo of growth in [discovered] geological reserves should be markedly greater than the tempo of development of the industry. At the same time, the task was established of a material increase in the thoroughness of prospecting of known geological reserves by transferring reserves of the C<sub>1</sub> and the C<sub>2</sub> Categories into higher categories.

To realize these goals large funds were invested in geological prospecting. As a result of the work done in just the first two years of the Second Five-Year Period, the copper reserves in the deposits were increased from 13.6 millions of tons (1932) to 16.95 millions of tons with an increase in the share of reserves of Categories A - B in the total reserves from 25.2 per cent (1932) to 47.7 per cent (1934). The lead reserves increased during the same period from 3.17 millions of tons to 4.46 millions of tons, and zinc from 6.58 to 8.8 millions of tons. In contrast with copper, the basic increase in reserves was accomplished here through a growth in reserves in Category C<sub>1</sub>, in which connection the share of reserves of the A - B Categories was decreased somewhat: from 29.2 to 25.8 per cent for lead and from 29.6 to 28.8 for zinc [2]. There was also an increase in the reserves of bauxite and nickel.

Along with a large increment in the reserves in previously discovered deposits, the large Blyava deposit of copper-bearing pyrites in the Orsk Region [2] and others. In the case of lead the reserves were increased by the mining areas of the Altay, especially in the Rider mining region, and also in the Tetyukhe region (Far East).

It should be noted that during the last three years of the Second Five-Year Period the increment of reserves slowed down.

The expansion of the raw materials base was accompanied by a simultaneous increase in capital investments in nonferrous metallurgy, including mining and ore dressing in the total.

About 1.5 billion rubles was invested in the nonferrous metals industry in the First Five-Year Period, but 4.0 billion rubles was to be invested in the Second Five-Year Plan.

Construction was continued on the Chimkent Lead Plant in the Kazakh SSR, the Chelyabinsk Zinc Plant, and the Dneprovsk Aluminum and Magnesium Plants in the Ukrainian SSR, and the construction of a number of new mines, concentration plants was carried out: the Severoural'sk bauxite mines, the Sredneuralsk and Balkhash Copper Smelters, the Blyava (Mednogorsk) Copper-Sulfur Plant, the Noril'sk and Monche-Tundrsk Copper-Nickel Combines, the large Leninogorsk Flotation and Ore Concentration Plants and a whole series of other mines and plants.

As a result of the completion of new construction during the Second Five-Year Period, the following were put into operation: the Chimkent Lead Plant and a number of mining and ore dressing enterprises in Southern Kazakhstan and the Central Asian Republics which supply it with raw materials; the Dneprovsk aluminum, magnesium, and electrode plants; the Chelyabinsk Electrolytic Zinc Plant; the Ufalety Nickel Plant (in the Urals); a large mining and ore dressing tin combine in Eastern Siberia; and a number of other large enterprises of nonferrous metallurgy.

the extraction of bauxite from the large Severoural'sk bauxite mines began in 1934.



The new mines and plants which were put into operation in the Second Five-Year Plan were greatly superior in capacity to the enterprises of pre-Revolutionary times. Thus, the 20 operating copper smelters produced a total of 32,000 tons of crude copper, but the seven copper plants in operation in 1937 had a total capacity of about 140,000 tons of copper. The same thing is true of the lead-zinc industry.

In order to raise the technological level in this branch of industry and the development of new progressive technological procedures and the complex processing of raw materials, a number of branch scientific research institutes were established: the State Scientific Research Institute for Nonferrous Metals (GINTsVETMET); the State Scientific Research Institute for Rare Metals (GIREDMET); and also institutes for planning: GIPROTsvETMET /Gosudarstvennyy institut proyektirovaniya predpriyatiy promyshlennosti tsvetnykh metallov - State Institute for the Planning of Nonferrous Metal Processing Plants/; GIPRONIKEL' /Abbreviation not known, presumably State Institute for the Planning of Nickel Processing Plants/, GIPROALYUMINIY /Abbreviation not known, presumably State Institute for the Planning of Aluminum Processing Plants/; and others.

In order to train skilled specialists for nonferrous metallurgy a number of special institutions of higher learning were established; in particular, the Moscow Institute of Nonferrous Metals and Gold imeni M. I. Kalinin; the Mining and Metallurgical Institute in Ordzhonikidze; a number of technical schools /TEKHNIKIMY/, and other training establishments.

By the end of the Second Five-Year Plan (1937), nonferrous metallurgy attained a level of annual production which exceeded the maximum pre-Revolutionary level in copper production by several times, and in lead and zinc production by several tens of times. In 1937 the smelting of copper reached 92,000 tons of electrolytic copper and 99,000 tons of crude copper, lead -- 63,000 tons, zinc -- 78,000 tons. The production of aluminum amounted to 37,700 tons against 900 tons in 1932 /1/.

Now branches of nonferrous metallurgy were established: aluminum, magnesium, nickel, tungsten, hard alloys, and others.

Along with the large quantitative growth in nonferrous metallurgy in the Second Five-Year Period, it was radically reconstructed technologically. In the field of ore dressing this period was marked by the construction of new large plants, basically mechanized flotation plants with the introduction of selective flotation; in the field of metallurgy - a transition to more progressive procedures in technological processes: reverberatory smelting in copper production, the electrolytic in place of the retort method of zinc production. In 1937 the share of the copper obtained in reverberatory furnaces reached almost 60 per cent, the relative share of zinc obtained by the electrolytic method was 51 per cent, that is, higher than in the United States (42 per cent).

The technological reconstruction in mining was shown in the mechanization of mining work, loading work, surface and subterranean hauling, water removal, lifting and other operations.

The technological reconstruction of nonferrous metallurgy led to a significant improvement in the basic technological-economic indices of the work of its enterprises, a large reduction in losses incurred during ore dressing and in metallurgical processes. Thus, the metallurgical recovery of copper in shaft /furnace/ smelting was raised from 73 per cent in 1932 to 90.4 per cent in 1937. For reverberatory smelting (in the copper smelters of the Urals) the recovery of copper rose, respectively, from 70 to 93.3 per cent. The daily production from a shaft furnace during the same period rose from 28.5 tons to 54.2 tons per square meter (cross section in the tuyere region). In reverberatory smelting the daily production of unroasted concentrate per square meter of hearth was also increased more than two fold--from 1.2 tons in 1932 to 2.3 tons in 1937.

Losses in the ore concentration plants and the metallurgical plants of the lead-zinc industry were markedly reduced.

However, even by the end of the Second Five-Year Plan, the nonferrous metallurgy of the USSR still did not satisfy the rapidly growing consumption of the nation, and in 1937 a considerable quantity of nonferrous metals was imported.

According to published figures [1] on foreign trade, 78,500 tons of different nonferrous metals were imported in 1935 at a total cost of 118.2 million rubles, 84,900 tons in 1936 for 156 million rubles, and 101,061 tons during the first nine months of 1937 for a total of 209.8 million rubles. In particular, in just the first nine months of 1937, the importation of copper amounted to 50,700 tons and lead 29,600 tons. Importations of zinc (2,887 tons) and aluminum (807 tons) were sharply curtailed in 1937. The nation was essentially freed from importation of zinc and aluminum.

The Balkhash and the Sredneuralsk Smelters were not put in to operation in the Second Five-Year Plan. The existing production capacity was far from being fully utilized: the losses of metal in processing were still large: and the polymetallic ores were not used in a sufficiently complex manner.

On the basis of the resolutions of the XVIII Party Congress, the Third Plan for the development of the National Economy and Industry (1938-1942) was established, and, in particular, a plan for the further development of nonferrous metallurgy.

On the field of the nonferrous metals industry the XVIII Party Congress set the task of increasing the production of nonferrous metals to such levels as would satisfy the rapidly growing consumption of the national economy and the national defense. In accordance with this, the Third Five-Year Plan provided that the production of crude copper in 1942 should be increased 2.8 times, and aluminum no less than 4 times (in comparison with 1937). A large increase in production was mapped out for lead, zinc, tungsten, and molybdenum.

In the Third Five-Year Plan for the development of nonferrous metallurgy a great deal of attention was devoted to the complex utilization of raw materials and reduction of losses in the processing of ores.

The Balkhash Copper Smelter and the Yuzhnoural'sk Nickel Plant was started to work in 1938.

In 1939 the Ural'sk Aluminum Plant, built on the basis of the Severoural'sk bauxite deposits, was put into operation: the Srednoural'sk Copper Plant and the Mednogorsk Copper-Sulfur Plant started work, and the very large new flotation plant at Leninogorsk and a number of other enterprises also started work.

The industry concerned with small-scale and rare metals saw a significant development in the period from 1938 to 1941.

In 1940 the USSR occupied one of the leading places in the world in the production of nonferrous metals. However, by the beginning of World War II the absolute level of production of nonferrous metals was inadequate and did not cover the growing requirements of the nation, its national economy and defense.

#### THE PERIOD OF WORLD WAR II

During the years of World War II nonferrous metallurgy developed under very severe conditions.

The development of the war industries increased the needs for nonferrous metals. Moreover, the production capacity of nonferrous metallurgy was decreased during the first years of the war.

the Dneprovsk and Volkhov Aluminum Plants were put out of commission, along with the Dneprovsk Magnesium Plant, the Severonikel' Northern Nickel Plant, the Tyrnyy Auz Tungsten-Molybdenum Combine, the Nikitovskiy Mercury Combine, the Lead-Zinc Combine in the Northern Caucasus, the Zinc Plant in Konstantinovka (Ukrainian SSSR), and others.

The large amount of new construction was undertaken in the eastern regions of the nation under severe war conditions. In a very short time, nonferrous rolling plants whose capacity surpassed the prewar level were built in the Urals and in Kazakhstan on the base of evacuated equipment and the maximum use of available production space in local enterprises.

During the war years great progress was achieved in the aluminum industry due to an increase in production capacity and improvement in the operation of existing equipment (Ural'sk Aluminum Plant). The raw materials and the power bases of the aluminum industry were markedly expanded, and the capacity of the Severoural'sk bauxite mines was increased. The construction of an aluminum plant in Western Siberia was started and the first stage of the plant was put into operation in 1943 (Stalinskiy Plant).



At the same time, a second large aluminum plant was built in the northern part of the Urals (Bogosloskiy) which began to produce metal on 9 May 1945. In the Kirgiz SSR a mercury combine built on the base of the Khaydarkan mercury deposits was put into operation.

The capacity for production of tungsten and nickel by the end of the war markedly exceeded the prewar capacity and the capacity of the tin mines and plants in the northeastern part of the USSR was markedly expanded.

The gross production of nonferrous metallurgy (including the extraction of ores) was greater in 1945 than in 1940, even though industry as a whole had declined in this period [4]. The production of aluminum, and magnesium, tungsten, and nickel had shown a special increase and the production of copper, lead, and zinc had declined.

After the XVIII Party Congress, geological prospecting projects were extensively developed in all the basic regions of the nation, but the plan for the conversion of known reserves into higher categories and the preparation of raw materials and extraction were not fulfilled, which hampered the development of the extraction of ores in 1941-1942.

The growth on the production of nonferrous metallurgy during World War II could not, of course, wholly satisfy the nation's growing needs for nonferrous metals during those years; and imports from abroad, particularly in articles made of nonferrous metals, reached significant proportions.

The development of nonferrous metallurgy during World War II was possible due to the large amount of capital construction in the eastern, rear regions of the nation and also due to the great amount of geological prospecting work in these regions.

Copper prospecting projects were carried out successfully in the Dzhezhazgan deposits and as a result sections of the deposit which were more rich in copper were discovered for priority processing. Many new deposits of a number of nonferrous and rare metals were discovered and, as a result of speedily conducted prospecting work, the reserves of tungsten and molybdenum were significantly increased (Kazakh SSR, Eastern Siberia), and of tin (the northeastern part of the USSR, and of bauxite (Western Siberia), and others.

As Soviet troops moved to the west and liberated temporarily occupied territories, ruined industrial enterprises were restored in comparatively short periods, including mines and plants belonging to nonferrous metallurgy in Ukraine, the Northern Caucasus, and in the Northwestern regions of the nation.

#### THE FOURTH FIVE-YEAR PLAN

After the victorious conclusion of World War II, the Party and the government established the task of restoring, during the next Five-Year Plan (1946-1950), not only those regions of the nation which had suffered during the war and regaining the prewar level of industry and agriculture, but also surpassing that level to a significant extent.

In the law of the Five-Year Plan for 1946-1950 were defined the tasks for the development of nonferrous metallurgy and the tempos and level of production in 1950.

In accordance with this plan, the production of copper should be increased 1.6 times in the Five-Year Plan, aluminum 2.0 times, and magnesium 2.7 times, lead 2.9 times, zinc 2.5 times, tungsten concentrate 4.4 times, molybdenum concentrate 2.1 times, tin 2.7 times, and nickel 1.9 times.

In order to ensure such a growth in production, the plan stipulated increasing the capacity of mines, concentration plants, and metallurgical plants in existing enterprises, and also the construction of a number of new enterprises.

In the copper industry the principal object of construction in this Five-Year Period was the Dzhezkazgan Combine (See Note As yet consisting of mines and ore concentration plants). In the lead-zinc industry the Five-Year Plan devoted special attention to the industrial development of the mountainous areas of Altay. The Five-Year Plan set large tasks for the aluminum industry and also the rare metals industry.

The capital investments in nonferrous metal industries were set at 12 billion rubles for 1946-1950 against 7.5 billion rubles (by plan) for the Third Five-Year Plan, 4 billion rubles for the Second Five-Year Plan, and 1.5 billion rubles for the First Five-Year Plan.

The Fourth Five-Year Plan was fulfilled by industry as a whole ahead of time--in four years and three months.

By the end of 1950 marked progress had been achieved in nonferrous metallurgy, too. As a result of the construction of new mines, ore concentrations plants, and also improvement in the work of operating enterprises, the production of copper, aluminum, nickel, lead, zinc, and other nonferrous and rare metals surpassed the pre-war level by significant amounts. The production of copper in 1950 surpassed the 1945 level by 83 percent, the production of zinc and lead was increased respectively by 2.32 and 2.38 times. The production of aluminum, magnesium, tungsten, and tin rose markedly.

The plan for production of nonferrous metals was not fulfilled for certain metals, for example, for lead.

The absolute levels of production achieved in 1950 were far from satisfying the needs of the nation, especially in respect to copper, lead, aluminum, and molybdenum. Therefore, the directives of the XIX Party Congress in regard to the Fifth-Five-Year Plan (1950-1955), like those in regard to previous plans, outlined a more rapid tempo in the growth of production of nonferrous metals as compared with a number of other branches of industry. With a general growth of 70 per cent in industrial production in the Five-Year Period, the following tempos of growth in production were outlined for nonferrous metallurgy: 90 per cent for copper, 2.5 times for zinc, not less than 2.6 times for aluminum, 2.7 times for lead, 1.8 times for tin, and 1.53 times for nickel

[5].

In the Fifth Five-Year Plan a great deal of attention was devoted to raising the technological level of production, the further mechanization of mining and laborious work, the automatization and intensification of production processes, increasing the complex use of ores and the extraction of all metals from them.

Figures on the development of the production of nonferrous metals are presented in Table 2.

TABLE 2

	1955, Per Cent of 1950 - 1955
Total production of nonferrous metallurgy	195.0
Which includes	
Refined copper	153.0
Lead	230.0
Zinc	200.0
Aluminum	280.0
Nickel	137.0
Tin	-----

[Translator's Note: The column heading is not clear, presumably means the increase from 1950 to 1955]. The plan for the production of aluminum in the Fifth Five-Year Plan was overfulfilled, the increase being 2.8 times instead of 2.6 times, but was not fulfilled in the production of certain other nonferrous metals.

Technological-economic indices were markedly improved in the Fifth Five-Year Period.

In many mines with underground methods for extracting ores, that is, more laborious methods, labor productivity was materially increased.

Indices of the increase in labor productivity are given in Table 3.

In a number of mines the specific expenditures of materials and the production costs per ton of ore were reduced and the percentage of depletion was decreased (for example, in the Degtyarsk Mine from 16.0 to 7.4 per cent). Mechanization of the basic production processes and loading onto railroad cars rose significantly.

In the Dzhezkazgan Mine, with complete mechanization of drilling, mechanization of hauling from cleared workings increased from 63 to 95.5 per cent in the Fifth Five-Year Period, that of removal of rock during the cutting of passageways from 34 to 83.5 per cent, underground hauling from 80 to 96 per cent and surface hauling from 79 to 98 per cent.

TABLE 3

Name of Mine	Labor Productivity, in Cubic Meters			
	Per Man Per Shift	Per Worker in Stope	Per Man Per Shift	Per Worker in Stope
	1950		1955	
Krasnoural'sk	0.28	1.17	0.36	1.58
Dzhezkazgan	0.90	2.21	1.53	4.85
Leninogorsk	0.46	1.04	0.66	1.38
Achisay	0.37	0.61	0.59	1.13

There are also important changes in the geographical distribution of nonferrous metallurgical enterprises, principally in a movement to the east. The shift began even in the Third Five-Year Period (1938-1942), was intensified during World War II, and achieved further intensive development in the postwar years.

Construction was carried out on a number of new large plants of the lead-zinc industry in the mining areas of the Altay (the Ust'-Kamenogorsk Polymetals Combine) and in Central Asia (on the base of the large Altyn-Topkan deposits): the tin industry is achieving further development in the Far East; new copper deposits are being opened in the Urals (Bashkir ASSR) and in Central Asia (Almalyk deposit). Aluminum production is being developed in the East.

The recovery of metals in ore dressing and in metallurgical processes was increased somewhat in individual nonferrous metallurgical enterprises in the Five-Year Period [6].

Reduction of the recovery indices in some ore concentration plants is explained by a deterioration in the quality of the raw material--partial use of oxidized and mixed ores along with sulfide ores.

The complex use of polymetallic and chalcopyrite ores was also increased.

In the Ust'-Kamenogorsk Polymetals Combine up to 12 valuable components are recovered from the ores in the form of pure metals and their semi-finished goods and sulfur in the form of sulfuric acid. In the "Elektortsink" Plant up to 11 components are recovered, in the Leninogorsk Plant up to 9 components. Many enterprises, including some copper plants in the Urals, extract 5-8 components. One should note in particular the successful mastery of the complex use of nephelites to obtain alumina, alkalies, and cement from them, also certain rare elements (Volkhov Aluminum Plant).

During the Fifth Five-Year Period the production of 20 new, in particular, rare metals: indium, germanium, thallium, tellurium, rhenium, cerium, niobium, and tantalum. Significant achievements were scored in the use of sulfur vapor from polymetallic ores.

The growth in recovery of metals, the better utilization of equipment, the reduction in specific expenditures of basic and auxiliary raw materials, also fuels, decreased the production costs of output. Thus, for example, the production costs per ton of crude copper produced in the Krasnoural'sk Plant was reduced almost 18 per cent in the Five-Year Period, in the Sredneural'sk by 29 per cent, and in the Karabash by 20 per cent. The cost of metallurgical production of lead in the Chimgent Plant was reduced by 21 per cent.

However, not all the reserves in increased production, reduced production costs, and increased technological-economic indices were utilized by the industry. Thus, in 1955 losses still remained significant in the processing of ores and concentrates. Losses of copper in all processes in 1955 amounted to more than 20 per cent of the copper contained in the ore, the losses of lead 33 per cent, and in many other metals considerably more [7]. In just a few plants in the Urals thousands of tons of zinc are lost every year in processing of ores one should mention the still low quality of lead, copper, and other concentrates. Thus, lead concentrates usually contain a total of only 45-47 per cent of lead and a great deal of admixtures. In the meantime, in the best foreign concentration plants the lead content in the concentrates amounts to 68-70 percent. This is true likewise of copper concentrates, especially from the Urals, in which the copper content does not exceed 12-15 per cent, while in the best foreign plants the metal content (copper) amounts to 35-40 per cent.

Thus, despite the large growth in the production of nonferrous and rare metals during the Fifth-Five-Year Period, the absolute level of production attained fell far short of satisfying the growing needs of the national economy, the capacity of operating enterprises was insufficiently utilized, and the losses of metals during processing of the raw materials were significant.

#### THE SIXTH FIVE-YEAR PERIOD

The directives adopted by the Congress of the KPSS /Kommunisticheskaya Partiya Sovetskogo Soyuza - Communist Party of the Soviet Union/ in respect to the Sixth Five-Year Plan for the Development of the National Economy of the USSR for 1956-1960 stipulated a new, mighty increase in the productive forces of the nation. Large and very responsible tasks were set for nonferrous metallurgy in the USSR.

In the field of nonferrous metallurgy, the Sixth Five-Year Plan specifies the following increases in 1960 over 1955: for refined copper about 60 per cent, aluminum 2.1 times, lead 42 per cent, zinc 77 per cent, nickel 64 per cent, molybdenum production 2 times, tungsten concentrates 57 per cent, and commercial magnesium 2.1 times. The production of titanium and the rare metals is to be expanded significantly. Special attention was devoted to increasing the output of nonferrous and rare metals of high purity to ensure the further development of electronics, radio engineering, and the production of heat-resistant alloys.

The directives in regard to the Sixth Five-Year Plan stipulate the wide introduction of high-production technological processes, the multi-stage concentration of ores, combination methods of ore dressing and hydrometallurgical processing of ores, roasting in the "boiling zone", electrothermal and autoclave processes, the use of oxygen in nonferrous metallurgy. In the mining industry it will be essential to increase the extraction of nonferrous metal ores in the five-year period by means of the more effective open-cut method by 2.2 times and to increase the underground extraction of ore markedly by systems with mass equipment. The recovery of metals from ores and concentrates should be increased materially, and more complex processing of raw materials should be organized. In this connection, scientific research should be intensified on searching out the most economical methods for winning nonferrous metals, also on further improvement and mastery of the technology of the production of rare metals.

On the basis of the rapid introduction of new technology, the productivity of labor should be increased by about 90 per cent with a growth in the productivity of labor for industry as a whole of not less than 50 per cent.

The Sixth Five-Year Period is a five-year period of new, large growth of capital investments in nonferrous metallurgy--about 2 times as compared with the Fifth Five-Year Period with a general growth in capital investments in industry as a whole of about 60 per cent.

The plan specifies increasing the capacity during the Sixth Five-Year Period for the production of aluminum (primary) by approximately 2.7 times, refined copper by 50 per cent, lead by 54 per cent, zinc by 1.8 times, nickel by 1.7 times, and sulfuric acid in the enterprises of nonferrous metallurgy by 5.2 times.

The Sixth Five-Year Plan stipulates maximum use of the internal resources of the operating of production and the use of the available production capacity should result in an increment of refined copper of 42 per cent, zinc 47 per cent, lead 34 per cent, aluminum 20 per cent, and nickel 60 per cent.

In accordance with the directives of the Congress of the KPSS in respect to geological prospecting work in 1956-1960, it will be necessary to ensure the following increases in prospected reserves: nickel 30-35 per cent; copper, titanium, and bauxites 40-45 per cent; niobium 50-55 per cent; lead and tin 55-60 per cent; and mercury 75-80 per cent.

Special attention is devoted in the directives of the XX Congress of the KPSS to the problem of ensuring more rational utilization of raw materials, fuels, electric power, and other material resources.

The first two years of the Sixth Five-Year Period provide evidence of successful implementation of the directives of the Congress of the KPSS in respect to nonferrous metallurgy. In this time significant achievements have been attained in equipment enterprises with new technology, further mechanization, automation and electrification of technological processes, and the extensive introduction of chemical methods.



In mining there has been a conversion to a more productive system of extraction by the open-cut method with maximum mechanization and automatization of all basic processes and the application of new, more highly perfected machinery: excavators with a capacity of 5-6 cubic meters, self-loading railroad cars with a load capacity of 40-50 tons, and electric locomotives with a weight of up to 150 tons, et cetera.

More powerful drilling machines are being used in the mines and removable boringbits tipped with hard alloys have been introduced. As a result of the introduction of new technology, open-cut works are effectively used even when it is essential to remove 7-10 cubic meters of rock for one cubic meter of ore [9].

Along with improvement and increased capacity of machinery used in ore dressing, methods for ore flotation are continually being perfected with the purpose of increasing the recovery of metals from the ores, obtaining the purest concentrates and intermediate products with their further processing through the use of special methods (VEL'TS-EVANIE /Translator's note: English equivalent not known. According to the Kratkiy Tekhnicheskiy Slovar' (Page 135) this is a sintering operation in which zinc or lead ores or wastes are heated in a rotary retort to a heat of 1300 degrees and the vapors are then passed through filters, pyroselection, and electrothermy). The application of multistage (Mirgalimsay Lead Ore Concentration Plant) and combined methods of ore dressing and hydrometallurgical processing of ores (Balkhash Copper Ore Concentration Plant), autoclave processes for processing complex molybdenum-tungsten products (Tyrny Auz Ore Concentration Plant), and also nickel products are being carried out.

The reagent industry has been markedly improved.

A number of new processes have been introduced into the metallurgy of the heavy metals which increase the productivity of available equipment of the degree of complex use of raw materials. Thus, for example, in 1957 almost all zinc plants have furnaces for roasting in the "boiling layer" which make it possible to increase the productivity of a roasting furnace by 4.5 times per square meter of area and almost 10 times per cubic meter of volume of the furnace with a sharp increase in labor productivity and concentration of sulfurous gases [5, 10].

In the copper and the lead industries, along with improvement and increased capacity of metallurgical equipment (reverberatory furnaces, converter equipment of large capacity), favorable conditions have been created for the recovery of rare metals (molybdenum, cadmium, selenium, tellurium, and cobalt). Electric smelting has been introduced in part (nickel, copper, lead).

In alumina production higher-capacity furnaces with a length of 120 meters have been introduced for sintering, also decomposers with a capacity of 1000 cubic meters and five-stage thickeners; in the production of aluminum, electrolytic cells of greater capacity and productivity with a capacity of 110-120 thousand amperes with top electrodes are being introduced in place of the present capacity of 50-60 thousand amperes.

Great progress has been made in expanding the raw materials base of nonferrous metallurgy. In the vicinity of the Turgay Depression of Kustanay Oblast' (Kazakh SSR) large reserves of bauxite, lead, titanium, copper, zinc, and antimony have been discovered. A new deposit of copper has been discovered in Chita Oblast'; bauxite, nickel, titanium, in the Ukrainian SSR; tungsten and molybdenum in Siberia and Nazakhstan [11], and large deposits of copper (Gaysk) and nickel (Buryktal) have been discovered in the southern Urals.

The progress achieved in the development of nonferrous metallurgy in the USSR ensures the further enhancement of its production capacity, qualitative indices, and reduction of production costs of the metals.

## NONFERROUS METALLURGY

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## II. THE SCIENTIFIC BASE. THE ORGANIZATIONAL STRUCTURE OF THE INDUSTRY.

The newly established scientific research institutes of nonferrous metallurgy have had to fulfill a large role in the mastery of the numerous processes used in the production of nonferrous metals.

Scientific and experimental studies were extensively developed in the State Scientific Research Institute of Nonferrous Metals (GINTSVETMET), which was organized at the end of 1929 on the basis of the nonferrous and rare metals laboratories of the All-Union Institute of Mineral Raw Materials (VIMS). In order to bring scientific research work closer to the sites of the construction of new enterprises, branches of GINTSVETMET [Gosudarstvennyy nauchno-issledovatel'skiy institut tsvetnykh metallov - State Scientific Research Institute of Nonferrous Metals] were established in Leningrad, Sverdlovsk, Irkutsk, and Tashkent.

Subsequently, independent affiliated institutes were organized on the base of these branches: VAMI [Vsesoyuznyy alyuminiyevo-magniyevyy institut - All-Union Aluminum-Magnesium Institute] in Leningrad; IRGIREDMET [Gosudarstvennyy institut redkikh metallov - State Institute of Rare Metals] in Irkutsk; UNIPROMED' [Ural'skiy nauchno-issledovatel'skiy institut mednoy promyshlennosti - Urals Scientific Research Institute of the Copper Industry].

GIREDMET [Gosudarstvennyy nauchno-issledovatel'skiy institut po redkim metallam - State Scientific Research Institute for Rare Metals] was separated from GINTSVETMET in Moscow. Research institutes were established for the nickel industry (GIPRONIKEL') [Abbreviation not known, apparently State Institute for the Planning of Enterprises of the Nickel Industry] in Leningrad and for the tin industry (TsNIIolovo) [Abbreviation not known, apparently Central Scientific Research Institute for the Tin Industry] in Novosibirsk.



In the post war period the GINTSVETMET branch in Ust'-Kamenogorsk was reorganized into the independent scientific research institute VNIItsvetmet [Abbreviation not known, apparently All-Union Scientific Research Institute for Nonferrous Metallurgy].

The organization of branches and bases of the Academy of Sciences, USSR in the large centers of the nation and the capitals of the union republics made it possible to bring science close to production and provide quick solutions to construction problems on the sites.

In many branches of the Academy of Sciences, USSR and also in the Academies of Sciences of the republics, special laboratories were established for the metallurgy of nonferrous and rare metals which were subsequently transformed into large scientific metallurgical centers of the nation. The Institute of Metallurgy of the Urals branch of the Academy of Sciences, located in Sverdlovsk, the center of the Urals, in which the construction of copper smelters, aluminum, nickel, and zinc plants was widely developed at a number of points and the Institute of Metallurgy and Ore Dressing of the Kazakh Academy of Sciences in Alma-Ata, capital of Kazakhstan, with the copper smelters, lead and zinc plants under construction in its territory, became centers for experimental studies for the expanding nonferrous metallurgy of the two of richest regions of the nation.

The organization of the Institute of Metallurgy of the Academy of Sciences, USSR in Moscow and the successful work in coordinating scientific activities in the field of nonferrous metallurgy of the institutes and the laboratories of the branches and Academies of Sciences of the republics have made possible more rapid and correct solutions of the most important problems of theory and production in nonferrous metallurgy.

In order to disseminate scientific and technical knowledge in the field of the metallurgy of nonferrous metals, and in order to facilitate the exchange of experience and to publicize the work of the plants and the institutes, the journal Tavetnye metally [Nonferrous Metals] was organized. The first number of the journal came out in January 1930.

The first number of the popular journal Opyt predpriyatiy [Experience of the Enterprises] was published at the beginning of the second half of that year. Subsequently, in the prewar period, the specialized journals Legkie metally [The Light Metals] and Redkie metally [The Rare Metals] came out.

In 1930, in the dawn of the development of Soviet nonferrous metallurgy, there were 346 engineers and 458 technicians in the entire nonferrous metals and gold industry. These included: engineers with production experience up to one year--74.3 per cent, up to three years --18.5 per cent, and more than three years -- 7.2 per cent. The requirements of the nonferrous metals and gold industry in respect to engineers was covered in 1930 by 32 per cent, and technicians by 24 per cent. The requirements of these branches of industry, on the other hand, in respect to engineers in 1930-1933 were determined to be about 11,000 persons.

The starting and the operation of the numerous nonferrous metallurgical enterprises stipulated by the plan could be ensured only on condition that the necessary engineering-technical personal be trained. Specialized establishments of higher training were established for this purpose: the Moscow Institute of Nonferrous Metals and Gold (1930) on the base of the Department of the Metallurgy of Nonferrous Metals of the Mining Academy; The Kazakh Mining-Metallurgical Institute in Alma-ata; the Eastern Siberian Mining-Metallurgical Institute in Irkutsk; and the Northern Caucasus Mining-Metallurgical Institute in Ordzhonikidze.

In addition, separate departments and chairs of metallurgy of the metallurgy of the nonferrous metals were functioning in the polytechnical institutes of Leningrad, Sverdlovsk, and Kiev.

A network of technical schools [TEKHNIKUMY] was set up to train specialists of intermediate skills was developed. The Industrial Academy in Sverdlovsk trained engineering-managerial and supervisory personnel up to the war.

Scientific research work was conducted on a wide front on the concentration of ores of the nonferrous metals, on pyrometallurgical and hydrometallurgical methods for processing ores and concentrates, and on the refining of metals in all the training establishments and in the laboratories of the specialized departments.

The fruitful work of the scientists of the higher training establishments was merged into a single stream with extensive scientific research projects carried on by research institutes and the plant laboratories.

It was necessary to supply the construction of the new enterprises of nonferrous metallurgy and the reconstruction of operating plants specified by the First Five-Year Plan with processed planning documentation. Up to the autumn of 1929, the planning of enterprises of nonferrous metallurgy was carried on by GIPROMEZ [Gosudarstvennyy institut proyektirovaniya metallurgicheskikh zavodov - State Institute for the Planning of Metallurgical plants].

The Decree [POSTANOVLENIE] of the Soviet of Labor and Defense of 2 August 1929 and the Order [PRIKAZ] of the VSNKh [Vysshiy sovet narodnogo khozyaystva - Supreme Soviet of the National Economy] established a special State institute--GIPROMEZ [Gosudarstvennyy institut po proyektirovaniyu predpriyatiy tsvetnoy metallopromyshlennosti - State Institute for the Planning of Enterprises of the Nonferrous Metals Industry]. Its mission included the maximum acceleration of the tempo of planning nonferrous metallurgical enterprises, particularly those projects whose construction should be started in 1930. Special attention was devoted to the quality of planning and the all-around use of foreign technical aid.

In order to obtain the most rapid mastery of the achievements of the American technology of that time, contracts were drawn up for technical aid with foreign companies and individual specialists, in particular with the UILER [Transliterated, possibly Wheeler] Office which had

a planning office in New York which dealt with the production of copper, lead, zinc, and nickel.

During the first years, the planning of all the plants of the nonferrous industry was concentrated in GIPROTsvETMET. Later, as the amount of planning projects was expanded and they became specialized, the GIPROALYUMINIY [Abbreviation not known, apparently State Institute for the Planning of Enterprises of the Aluminum Industry] Institute was established to make plans for aluminum and magnesium plants; SOYUZ-NIKEL'OLVOPROYEKT [Abbreviation not known, apparently All-Union Institute for the Planning of Enterprises of the Nickel and Tin Industries] was set up to make the plans for the nickel and the tin industries, later renamed GIPRONIKEL' [Abbreviation not known, apparently State Institute for the Planning of Enterprises of the Nickel Industry]; GIPROREDMET [Gosudarstvennyy institut po proyektirovaniyu promyshlennyykh predpriyatiy redkikh metallov - State Institute for the Planning of Rare Metals Industry Establishments] for enterprises of the rare metals industry; and GIPROZOLOTO [Gosudarstvennyy institut po proyektirovaniyu predpriyatiy zoloto-platinovoy promyshlennosti - State Institute for the Planning of Gold and Platinum Industry Establishments].

The brisk development of the nonferrous and rare metals industry required, even in the First Five-Year Period, suitable organizational forms for daily guidance and the operative solution of the many problems of planning and production. By decree [POSTANOVLENIE] of the Soviet of Labor and Defense, the Administration for the Management of Enterprises of Nonferrous Metallurgy, which was under the Supreme Soviet of the National Economy, and the Administration for the Management of Enterprises of the Gold Industry, which was under the people's Commissariat of Finance, were combined by the end of 1929 into the unified administration [UPRAVLENIE] TsvETMETZOLOTO [Vsesoyuznoye ob"edinenie po dobyche, obrabotke i realizatsii tsvetnykh metallov, zolota i platiny - All-Union Association for the Extraction, Processing, and Sale of Nonferrous Metals, Gold and Platinum].

Later, in order to bring management closer to the operating enterprises and those under construction, TsvETMETZOLOTO was divided by the decree [POSTANOVLENIE] of the Soviet of Labor and Defense of 7 August 1931 into three associations [OB"YEDINENIYA]: SOYUZALYUMINIY [Abbreviation not known, apparently All-Union Association of the Aluminum Industry] (Moscow), YuZhTsvETMET [Abbreviation not known, apparently All-Union Association of the Nonferrous Metals Industry of the Southern Regions] (Alma-Ata), and TsvETMETZOLOTO [Abbreviation not known, apparently All-Union Association of the Nonferrous Metals and the Gold Industries] (Sverdlovsk); and two trusts: ZAKTsvETMET [Zakavkazkiy gosudarstvennyy trest tsvetnykh metallov - transcaucasian state trust of Nonferrous Metals] (Tbilisi), and SEVKAVTsvETMET [Abbreviation not known, apparently North Caucasus State Trust of Nonferrous Metals] (Ordzhonikidze). YuZhTsvETMET contained all operating enterprises and those under construction in South Kazakhstan, Kirgizia, Uzbekistan,

Tadzhikistan, and Turkmenistan. The trusts SOYUZREDMET [Gosuderstvennoye vsesoyuznoye ob'edinenie redkikh elementov- State All-Union Association for Rare Elements] and SOYUZOLOVO [Abbreviations not known, apparently State All-Union Association of the Tin Industry].

Territorial associations and trusts of the nonferrous and the gold industries established, also scientific and planning institutes which had been established were subordinated by the decree [POSTANOVLENIE] of the Presidium of the VSNKh SSSR of 12 November 1931 to the newly organized GLAVTSVETMETZOLOTO [Glavnoye upravlenie po tsvetnym metallam, zolotu i platine - Main Administration for Nonferrous Metals, Gold, and Platinum] under the VSNKh SSSR.

With the reorganization of the VSNKh SSSR into the NARKOMTYa-ZhPROM [Narodnyy Komissariat tyazheloy promyshlennosti - People's Commissariat of Heavy Industry], GLAVTSVETMETZOLOTO became one of its production administrations.

However, a single administration for the whole nonferrous metals and gold industries turned out to be very cumbersome, and by order [PRIKAZ] of the People's Commissariat of Heavy Industry of 27 April 1933, GLAVTSVETMETZOLOTO was divided into two independent administrations: GLAVTSVETMET [Glavnoye upravlenie po dobyche i obrabotke tsvetnykh metallov - Main Administration for the Mining and Processing of Nonferrous Metals] and GLAVZOLOTO [Glavnoye upravlenie zoloto-platinovoy promyshlennosti - Main Administration of the Gold and Platinum Industries].

YUZhTsVETMET was liquidated. The new construction projects which had been under it: Dzhezkazgan, KAZPOLIMETALL [Kazakhskiy Polimetallicheskiy Kombinat - Kazakh Polymetals Combine], ALMALYKSTROY [Gosudarstvennoye upravlenie po stroitel'stvu Almalykskogo mednogo kombinata - State Administration for the Construction of the Almalyk Copper Combine], PRIBALKHASHSTROY [Abbreviation not known, possibly State Administration for the Construction of the Balkhash Copper Combine], were subordinated directly to the GLAVTSVETMET of the NARKOMTYa-ZhPROM.

The rapid growth of different branches of the nonferrous metals industry, the specific requirements of geological prospecting and mining work, and technological peculiarities in the production of different metals brought up the question of reorganization of GLAVTSVETMET into the independent People's Commissariat of Nonferrous Metallurgy with production administrations for copper (GLAVMED') [Glavnoye upravlenie mednoy promyshlennosti - Main Administration of the Copper Industry], for lead and zinc (GLAVSVINSTsOVOTsINK) [Abbreviation not known, apparently Main Administration of the Lead and Zinc Industries], for aluminum and magnesium (GLAVALYUMINIY) [Glavnoye upravlenie aluminievoy promyshlennosti - Main Administration of the Aluminum Industry], for nickel and cobalt (GLAVNIKEL') [Abbreviation not known, apparently Main Administration of the Nickel Industry], for tin (GLAVOLOVO) [Abbreviation not known, apparently Main Administration of the Tin Industry], and for rare Metals (GLAVREDMET) [Glavnoye upravlenie redkikh metallov - Main Administration for Rare Metals]. Enterprises of the gold and platinum industry, which had been unified in the

GLAVZOLOTO, were also transferred to the People's Commissariat of Non-ferrous Metallurgy. Later, the People's Commissariat was renamed the Ministry of Nonferrous Metallurgy. Repeated attempts to merge the enterprises of ferrous and nonferrous metallurgy into a unified People's Commissariat or Ministry failed to achieve favorable results. In comparatively short periods of time the ferrous and nonferrous branches of the metallurgical industry would be separated once more.

Implementation, in accordance with the resolutions of the Party Congress, of the reorganization of the management of the industry on the basis of establishing territorial Soviets of the National Economy will make possible the further development and growth of nonferrous metallurgy.

### III. THE DEVELOPMENT OF THE PRODUCTION OF SEPARATE METALS - COPPER

Copper production in pre-Revolutionary Russia was concentrated in four regions: in the Urals, the Caucasus, in the Kirgiz Steppe, and Minusinsk Kray.

The beginning of the copper industry in the Urals goes back to the 17th Century. The first copper smelter, the Pyskarskiy on the Yayva, was built in 1640. The rich copper ores (oxides), which contained malachite, copper lazurite, and cuprite, were smelted in simple blacksmith's forges. Two other copper smelters, the Polsvskiy and the Byysk, were built in the beginning of the 18th Century. In the second half of the 18th century the Bogoslovskiy and the Nikolo-Pavdinskiy Plants were working on the base of the Turinsk Mines. At the beginning of the 19th Century the annual production of copper amounted to 3000 tons. By the middle of the 19th Century, the copper of the Urals was being produced in a number of small plants. They smelted the rich oxide ores taken from the Mednorudnyansk, Gumeshevskiy, Pyshma, Turinsk, and other mines. In the period of the 18th - 19th Centuries, the copper industry achieved marked development on the base of the Perm sandstone.

Stroganov, Demidov, and other entrepreneurs built plants in the Perm, Kazan'-Vyatka, Orenburg, and Ufa Areas, smelting the ore from sections which were enriched into copper. As these sections became depleted, the exploitation of the tiny plants became unprofitable and the enterprises were closed. The Yugovskiy Plant which had been built in 1732 and ceased to be active in 1903 remained the longest in operation. A hydrometallurgical apparatus for leaching copper out of the oxide ores from the Pyanoborsk Deposit with a sulfuric acid solution was installed in 1892 in the Bondyuzhskiy Plant, in which copper had been smelted out of sandstone as early as the 17th Century. At the end of the 18th Century copper production had also sprung up in the Caucasus and in Minusinsk Kray.

At the end of the 19th Century the annual production of copper reached 6400 tons. In 1913, before the beginning of the First World War, 20,500 tons of copper were produced.



Up to the end of the 19th Century, copper metallurgy was based on the processing of oxide ores by a method of reduction smelting. As the upper oxide zones of the deposits which were rich in copper became depleted and the mines became deeper, sulfide ores were discovered which required other methods of processing. They began to apply to them the method of pyrite smelting of copper-pyrite ores in water-jacket furnaces which was being used in foreign technology. In such smelting of lumps of pyrite ores, the heat needed for the process was obtained by oxidizing the iron sulfides with the oxygen contained in the air blown in through tuyeres. The ferrous oxide formed in the smelting process was slagged by silica while the copper was concentrated in the form of matte or an alloy of cuprous sulfide and iron sulfide. They also began to apply the Bessemer process to copper matte in converters in the presence of silica, with all the iron going into the slag and obtaining pure copper sulfide, or fine Matte [Translator's note: Apparently an intermediate product]; it was then passed through a converter to obtain crude copper. In 1913, before the outbreak of World War I, six copper smelters were in operation in the Urals (Bogoslovskiy, Kalatinskiy, Pyshma-Klyuchovskiy, Polevskiy, Karabash, and Tanalslyk-Baymak), which operated with the pyrite or semipyrite methods of smelting. The transition to such smelting marked the second period of development of copper production in Russia, when they began intensified prospecting of chalcoppyrite deposits, both in the Urals as well as in other regions of the nation.

Electrolytic refining of crude copper was carried on in the Kysh-tym Copper Electrolytic Plant.

During World War I the production of copper in Russia gradually declined into a ruinous state and ceased entirely in 1918.

After the Civil War ended, in the restoration period, copper smelters were gradually put into operation again. In May 1922 the Kalatinskiy Plant was put into operation, and in 1924 the Tanalslyk-Baymak and the Alaverdy Plants. In 1925 the Karabash Copper Smelter, which was the largest plant at that time, was restored and put into operation. The raw material for all the restored smelters was chalcoppyrite ores which were processed by pyrite or semipyrite smelting in water-jacket furnaces which produced copper matte. When it was bessemerized in converters crude [Blister] copper was produced, which was refined on the spot in reverberatory furnaces, to so-called refining hearths. Up to 1910, shaft furnace pyrite or semipyrite smelting of chalcoppyrite ores with a consumption of coke with limits of 3-10 per cent was the sole industrial method for producing copper. The average copper content then had to be not less than 2 per cent. The limited reserves of chalcoppyrite ores in nature, the irreversible losses of all the iron and a large part of the sulfur during smelting, and the presence of large deposits of copper-containing porphyry ores with a relatively low copper content which could not be processed in shaft furnaces -- all this stimulated the development of a new technological process which

would include preliminary concentration of the ores, with subsequent smelting of those copper concentrates in reverberatory furnaces to matte and slag. During the period 1910-1925, world technology of copper production was distinguished by an exceedingly rapid introduction of this new method of winning copper. Thus, in the United States the use of shaft furnaces had already been wholly discontinued by the time of the world crisis in 1929. This made possible the progress in flotation concentration of ores and the mastering of the process of smelting the concentrates in reverberatory furnaces.

The Korsak-Pay Smelter was the first plant in the Soviet Copper industry to start working with the new method. The copper quartz ores of Dzhezkazgan were subjected to flotation concentration and concentrates containing 30 and more per cent copper were obtained. The concentrates were smelted in a reverberatory furnace to matte (50 per cent copper and more) and acid slag. The matte was then put through converters to obtain crude copper.

The Korsak-Pay Smelter was the first successfully to overcome the technical difficulties of smelting low-sulfur and low-iron concentrates to matte with a high copper content and high-silicon slag without adding sulfur-iron fluxes; the difficulties of bessomerizing rich matte were also overcome.

The construction of the Korsakpay Copper Smelter, which is located in Central Kazakhstan close to the Dzhezkazgan Copper ore Deposit, was begun even before World War I by concessionaires, but the work had not been completed. Construction of the smelter was completed and it was put into operation at the end of the restoration period, in 1927. The technological lay-out of the Korsakpay Smelter was the basis for the planning and construction of the largest copper smelters (combines) in the Soviet Union -- the Balkhash and the Dzhezkazgan.

In 1926 construction was started on the Krasnoural'sk Copper Smelter with a planned capacity of 20,000 tons of crude copper per year. Its plan was first to specify, on the basis of processing chalcopyrite ores, flotation methods of concentration which produced copper and pyrite concentrates. The copper concentrates were smelted in a reverberatory furnace. The pyrite concentrates were sent to sulfuric acid plants.

The Krasnoural'sk Smelter produced its first copper in the beginning of 1931.

The successful mastery of methods of selective flotation of the chalcopyrite ores from the Urals which resulted in production of copper and pyrite concentrates served as the starting point for extensive introduction of this method throughout the entire copper industry of the Urals. All achievements in the floatation of chalcopyrite ores and the smelting of concentrates in reverberatory furnaces were utilized in the new Sredneural'sk Copper Smelter.

The reconstruction of the Karabash and the Kirovograd Plants, which had made extensive use of shaft furnace smelting of chalcopyrite ores in water-jacket furnaces, was carried out in the direction of

developing floatation methods for concentrating copper and copper-zinc pyrite ores, with separation of copper and zinc concentrates. The smelting of copper and copper-zinc concentrates in reverberatory furnaces and obtaining matte and iron slag. [sic, presumably misprint. Apparently iron silicate slag], was mastered.

It must be noted that shaft furnace smelting of lumpy ores in water-jacket furnaces, which were suitably reconstructed then left in these smelters, continued to exist along with the development of reverberatory smelting. Combining shaft and reverberatory smelting made it possible, after sorting the ore, to send the lumpy portion to the water-jacket furnaces and the fine ore to the concentration plant. Lack of fine ore particles in the shaft furnaces markedly improved the technical performance in smelting. It became possible to process the dust from the converters and the reverberatory furnaces along with the concentrates in the reverberatory furnaces. Smelting with water-jacket furnaces was retained in the Tanalyk-Baymak Smelters. The Caucasian plants were wholly reconstructed and converted to the new method of work -- flotation concentration of the ores and selective separation of copper concentrates to be sent to reverberatory furnaces for smelting.

At the end of the restoration period, six copper smelters were in operation in the USSR. In 1929-1930 about 30,000 tons of copper was produced, thus the prewar level of production was surpassed (refer to table).

Smelter	Deposit, Region	Annual Production (Tons)
Kirovograd	Urals	11,370
Karabash	Urals	10,800
Korsak-Pay	Kazakhstan	5,480
Tanalyk-Baymak	Bashkiria	3,420
Alaverdy	Armenian SSR	2,250
Zangezur	Armenian SSR	820

In the middle of 1933, planning was started for a copper combine based on the newly-discovered Blyava deposits of chalcopyrite ores. The plant was planned and built in accordance with a Norwegian layout: cuprous pyrites mixed with 10 per cent coke dust were smelted in a water-jacket furnace with a closed top to copper matte, slag and sulfur vapor, which when condensed in waste-heat boilers, produced liquid sulfur.

The copper industry was developed intensively in the three pre-war Five Year Periods. These operating copper smelters were completely reconstructed and expanded: the Karabash, the Kirovograd, and the Alaverdy. The Irtysh Plant was put into operation. Two new, modern copper smelting combines were built -- the Sredneuralsk and the Balkhash.



The Sredneuralsk Copper Smelting Combine, located near Revda, is one of the larger enterprises of the copper industry of the Urals. The raw materials base of the smelter is the large Degtyarinsk Deposit of chalcopyrite ores with an average copper content of about 1.5 per cent. The ores also contain zinc, arsenic, and gold. They are subjected to flotation concentration to separate copper and pyrite concentrates. The technological layout for processing the copper concentrates consists of the following operations: roasting the copper concentrates in multihearth roasters; smelting the roasted concentrates in a reverberatory furnace on a coal dust fuel; bessemerizing the matte in converters; and pyro-refining the crude copper in a flame [reverberatory] furnace. Provision has been made for the use of the sulfur gases from the roasting furnaces and converters in making sulfuric acid. Steam boiler facilities were planned to make use of the heat from the waste gases from the reverberatory furnaces.

The Balkhash Mining and Metallurgical Combine, which rises up from the desert shores of Lake Balkhash, and operates on a base of the Kounrad Deposit of copper porphyry ores, is one of the largest copper smelting enterprises in the Soviet Union with advanced equipment. The first planning specifications for the construction of the copper smelter in Central Kazakhstan were drawn up by GIPROTsvETMET in 1930. It was specified that the plant would be located in the vicinity of Nura Station, 500 kilometers from the Kounrad Deposit. Later, the problem of the location of the plant was reexamined inasmuch as the copper smelter should receive chiefly copper concentrates from the concentration plant which would process copper porphyry ores from the Kounrad Deposit, whose reserves had been determined to be sufficiently large by that time. It was then decided to build the smelter and the concentration plant on the northern shores of Lake Balkhash, 18 kilometers south of the Kounrad mine. The plans specified pyrometallurgical processing of the sulfide concentrates, with smelting in reverberatory furnaces and hydrometallurgical processing of the oxide ores by leaching out with sulfuric acid solutions. The capacity of the pyrometallurgical plant was planned for 50,000 tons of copper per year, with a possible expansion to 100,000 tons; the hydrometallurgical plant was planned for an output of 15,000 tons of copper per year, with an 80 per cent recovery of metal. The planning specifications were approved in November 1930. However, the construction of the hydrometallurgical plant was not carried out later because the recovery of copper from the concentrates had reached 75-80 per cent due to advances made in the flotation of oxide ores. Since the oxidized and mixed ores made up the upper zone of the deposit and was worked only during stripping work, they were sent directly to the ore concentration plant.

At the same time modern copper smelters, operating on the layout of flotation of copper ore and smelting the concentrates in reverberatory furnaces, were being built, the old water-jacket furnaces were reconstructed, the technology of the process was improved, tending the

machinery was mechanized, and better technical-economic indices were achieved.

The installation of a coal pulverizing apparatus by each reverberatory furnace, forced delivery of secondary and tertiary air through burners, and use of the heat of waste gases under high-pressure steam boilers made it possible to obtain an increase in the productivity of the furnaces with a reduction in the specific fuel consumption per unit of melted charge.

The use of suspended magnesite crowns lengthened the campaigns [runs] of reverberatory furnaces.

Preliminary drying and roasting of high-sulfur copper concentrates, painstaking charging of materials and loading with hot charges have also contributed to increased productivity of reverberatory furnaces and improved technical-economic indices of the process.

By the beginning of World War II, the soviet Union already possessed a highly developed and mighty copper industry located within the depths of the nation, far from the front. By mobilizing internal resources, eliminating bottle-necks in individual enterprises, and perfecting the technology of production, the workers and the engineers put a great amount of effort into satisfying the requirements of the war period for copper.

The Kyshtym Electrolytic Copper Plant which was wholly reconstructed before the war, was new and one of the largest in Europe; the Pyshma Electrolytic Copper Plant, and also the electrolytic copper departments in the nickel plants which process copper-nickel ores completely supplied the nation with electrolytic copper.

In the postwar period, during the Fourth and Fifth Five-Year Periods, the improvement of technology and techniques of production was continued. As a result of improved mixing of the charge, a more even of materials to the smelting facilities, and rational combustion of fuels, the specific melts in reverberatory and shaft furnaces increased steadily. Large experimental projects on automatization of tending reverberatory furnaces yielded favorable results.

Preparatory work was carried out for new construction.

Construction was started on a metallurgical enterprise on the base of the mighty Dzhezkazgan Deposits which should consist of an ore concentration plant and a copper smelter; in the meantime the copper concentrates obtained from the Dzhezhazgan Ore Concentration Plant are hauled to the Balkhash Plant. Large deposits of chalcopryite ores discovered at Sibay (Bashkir SSR) have become the base for a new ore concentration plant which sends copper and copper-zinc concentrates to the copper smelters of the Urals.

Construction of the Almalyk Copper Enterprise which is to consist of mines, an ore concentration plant, and a copper smelter predetermines the development of the copper industry in Uzbekistan. In the ore concentration plant being built at Almalyk, the dressing of oxide and mixed ores is done by a combination hydrochemical method which combines in a single apparatus the processes of leaching out copper with

a sulfuric acid solution and cementation of copper with a finely crushed metallic iron and flotation of cemented copper. According to the planning, the copper concentrates obtained should be processed by electro-thermal smelting. The use of this scheme in the industry will open the way for the use of analogous ores from the deposits of Boschekul', Pirodoudan, and others.

Scientific experimental work on intensification of copper production is leading in the direction of roasting copper sulfide concentrates in the boiling layer; cyclic smelting of the copper charge; autothermal smelting of copper sulfide concentrates with an oxygen blast; the use of air enriched with oxygen in smelting with water-jacket furnaces and the bessemerizing of copper matte; to using a continuous process in the bessemerizing of matte; and the electric smelting of copper concentrates.

#### IV. MODERN SCIENTIFIC AND TECHNICAL PROBLEMS OF SOVIET NONFERROUS METALLURGY

In the 40 years of development of Soviet nonferrous metallurgy, the technological layouts which formed the bases of new construction projects were gradually improved in exploitation; the efforts of scientific research workers and innovators brought about constant improvements in the design of equipment and intensified individual operations.

Processes were mechanized and automatized, the specific (per unit of smelted metal) consumption of materials and power was reduced, and the productivity of labor was increased. The percentage of recovery of metals and their quality improved continually.

At the same time, as a result of studies of an exploratory nature which were carried on in scientific research institutes, laboratories, and in pilot facilities, a large amount of theoretical and experimental material was accumulated which predetermines the way for further development of the metallurgy of nonferrous metals on a new and higher technological level.

The directives of the Congress of the KPSS in respect to the Sixth Five-Year Plan for the Development of the National Economy of 1956 - 1960 point out that the Soviet Nation now has at its disposal all the prerequisites for solving, on the paths of peaceful economic competition, the basic economic task of the USSR in the historically shortest time -- that of catching up with and surpassing the most highly developed capitalist countries in per capita production.

Fulfillment of this task in nonferrous metallurgy should be based on observation of the two basic propositions:

Losses of metal in all stages of technological reduction -- in extracting the ores and bringing them to the surface, in the processes of ore concentration, in the metallurgical operations of processing the ores and concentrates with winning crude metals and in their subsequent refining -- should be minimal.

Complex processing of polymetallic, multicomponent ores should be observed -- all valuable components of the raw ores should be completely recovered in the form of production ready for consumption.

The positive solution of these problems, with continual establishment of essential sanitary and hygienic conditions of socialistic labor, presupposes uninterrupted improvement and intensification of existing processes, searches for new technological procedures and high-production equipment, and frequent radical changes in the processes for winning metals.

In order to ensure the planned gigantic growth in the mining of nonferrous metals in the near future it will be necessary to carry out a great amount of work; to make an extensive survey of the reserves in the ground, to map the boundaries of deposits, to select the most effective methods of mining, to prepare the delivery of ore to the surface, and to carry out its beneficiation and the processing of the concentrates obtained into metals and alloys.

It is necessary to take into consideration the specific nature of the economics of nonferrous metallurgy. The geological features of deposits of the ores of nonferrous metals are: the scattered nature of the small (as compared with iron ores) thickness of the beds; the varied nature of the structure of the beds which are found in the form of small lenses, veins, blocks, et cetera, which complicates the choice of a rational system for mining and the delivery of ore to the surface. The ores of nonferrous metals are distinguished by the complexity of their mineralogical composition. Their polymetallic nature and their low content of individual metals constitute a peculiarity of many ores. In many polymetallic ores the value of the accompanying metals, in particular rare and noble metals, frequently surpass the value of the principal metals several fold. This gives rise to the urgent need to rationalize technological processes with the purpose of recovering the accompanying metals.

The most serious attention should be directed to the possibility of concentrating accompanying metals from the group of rare and scattered elements in different intermediate products of production. In this connection, dust catching and refining of ferrous metals becomes exceedingly important, inasmuch as the dust and the wastes of production are collectors of a number of very valuable elements.

It is essential to make the maximum effort to overcome the difficulties connected with the complication of technological procedures for processing ores with many components.

Maximum electrification of the processes for winning and refining metals is one of the main ways for the progressive development of nonferrous metallurgy.

Use of electric power in the metallurgy of nonferrous metals is possible for electrochemical purposes (direct current) and for electrothermal purposes (alternating current).

The application of direct current in refining ferrous metals and the electrodeposition of metals from aqueous and molten media has acquired wide use in the metallurgy of nonferrous metals. Almost all the crude copper, bismuth lead, crude nickel produced, part of the tin are subjected to refining by electrolysis in aqueous electrolytes. Aluminum is partially subjected to electrolytic refining with the application of a molten electrolyte. Electrochemical methods for refining crude metals permits obtaining cathode metals of high purity with simultaneous concentration of valuable admixtures in the anode product. Thus, in the electrolytic refining of copper, the anode slime contains noble metals, selenium, and tellurium; in the refining of lead -- silver and bismuth; in refining nickel -- the platinoids, gold, silver, selenium, and tellurium; in refining aluminum -- gallium.

Electrolytic methods for separating metals from aqueous solutions and from molten forms are used in the metallurgy of a number of metals. The overwhelming portion of the world production of zinc is obtained by the electrolysis of an aqueous solution of its sulfate. About 20 per cent of the world production of copper is obtained by electrolysis of an aqueous solution of copper sulfate. In aluminum production, the electrolysis of aluminate in molten cryolite is the principal technological process. Magnesium is obtained chiefly by electrolysis of molten chlorides. The electrochemical method of processing matte metals is worthy of attention. With anode dissolution of copper-nickel-cobalt sulfide alloys or matte in sulfuric acid solutions, all three metals enter into the electrolyte with simultaneous cathode deposition of metallic copper. Noble metals and elementary sulfur remain in the anode slime. The cobalt is then separated from the nickel-cobalt electrolyte in the form of the hydroxide by oxidizing with ozone. Nickel is separated electrolytically from the purified solution of nickel sulfate in baths with diaphragms and an insoluble anode. Depositions of cobalt hydroxide are dissolved in sulfuric acid. The cobalt is deposited electrolytically from the solution in the form of a pure cathode metal.

With anode dissolution of copper-lead-zinc sulfide alloys or matte in aqueous solutions of sulfuric acid, the copper and zinc go into the electrolyte with simultaneous deposition of the copper on the cathode. Noble metals, lead sulfate, and elementary sulfur remain in the anode slime. Zinc is deposited electrolytically from the purified zinc solution.

The electrolysis of the sulfides of metals in molten media with winning of metals and elementary sulfur is a prospective trend in technical electrochemistry.

The development of technical electrochemistry in nonferrous metallurgy should be directed toward the study of anode and cathode processes with the purpose of decreasing the specific consumption of electric power per unit of metal and also of improving the quality of the cathode deposits.

A considerable portion of the heavy nonferrous metals is smelted in shaft and reverberatory furnaces with the consumption of large masses of air used in combustion of the fuel. At this time, in view of the volatility of large portion of nonferrous and rare metals and their chemical compounds at the high temperatures and the high velocities of the streams of gas, a significant carrying away of metals is observed. Therefore, as a rule, all metallurgical aggregates should be equipped with effective facilities for gas scrubbing and dust catching and also for regeneration of the heat of waste gases.

The application of electric furnaces in the smelting of ores and concentrates which contain nonferrous and rare metals permits a definite rationalization of the smelting process. Undoubted advantages of electric smelting aggregates include automatic regulation and rapid reaching of required temperatures, lack of fuel gases, and the possibility of mechanization and hermetization. Electrothermal methods can be applied to the smelting of copper and copper-nickel sulfide ores and concentrates to matte; in smelting oxide nickel ores to matte or nickel-iron alloys; in reduction smelting of lead agglomerates; in the continuous distillation of zinc from roasted zinc concentrates; in the smelting of oxide nickel ores to matte or nickel-iron alloys; in reduction smelting of lead agglomerates; in the continuous distillation of zinc from roasted zinc concentrates; in the smelting of roasted copper-lead-zinc sulfide products to metal, matte, and vapor phases; in the reduction smelting of cassiterite to tin; nickel and cobalt oxides to metals; cyanides to aluminum-silicates, etc cetera.

The smelting of nonferrous metals in resistance electric furnaces with the use of slag as a heating body gives rise to a number of technical problems whose solution will require corresponding theoretical and experimental studies. Study is required of the chemism of non-equilibrium reactions in systems of a metal-sulfur--oxygen--metals--oxygen--carbon, the conditions of dynamic equilibrium between slag, matte, and metal, electrical conductivity and the viscosity of slags, the mechanism and kinetics of processes which are going on in electrothermal furnaces.

New prospects in the metallurgy of metals obtained at the moment of reduction in vapor state open the way for the application of electrothermal furnaces which heat with high-frequency induction currents.

The use of electrothermal processes with vacuum (vacuum metallurgy) or high pressures (autoclave metallurgy) is very promising. The use of air enriched with oxygen or pure oxygen for metallurgical and thermal purposes is of very great importance in intensifying many metallurgical processes.

The rate of the processes of zone roasting of sulfide products in furnaces and in agglomeration machines, of bessemerizing of matte, of the combined process of roasting and smelting of copper sulfide concentrates in suspension in reverberatory furnaces is largely determined by the initial concentration of oxygen in the gas mixture and attains maximum values when pure oxygen is used.



The use of air enriched with oxygen in shaft furnaces not only makes it possible to increase the temperature in the focus of the furnace, but also increases the initial concentration of carbon monoxide in the gas mixture, thus making possible a more rapid course of reduction reactions.

## THE METALLURGY OF THE HEAVY NONFERROUS METALS

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The metallurgy of the heavy nonferrous metals has passed along a great path of development during the existence of the Soviet State. It has been transformed from a small, weakly-developed branch of production with artisan technology into a mighty industry equipped with new technology. Our country now produces all the necessary metals. Soviet nonferrous metallurgy occupies second place in the world in respect to scale of production.

This article, which examines the metallurgy of the principal heavy nonferrous metals: copper, nickel, lead, and zinc, will contain a brief elucidation of the winning of these metals by metallurgical processes.

One may mark the following four periods in the development of Soviet nonferrous metallurgy.

1. The restoration period which begins in 1918 and extends to 1925 - 1927. The principal task of this period was to restore normal work in the basic enterprises of the copper, lead, and zinc industries which had existed before the Revolution.

2. This period extended from 1925 - 1927 to 1941. The tasks were established of the reconstruction of old mines and plants and the construction of new enterprises, also the organization of the production of new metals hitherto not produced in the USSR, for example, nickel, also cadmium, bismuth, and other accessory metals which accompany the heavy nonferrous metals.

3. During World War II it was essential in the first instance to support the level of production already attained, then, so far as possible, to bring it to the size determined by war time requirements.

4. In the period after the war and at the present time, the main task consists in the most rapid possible restoration of the nonferrous metallurgical enterprises which had suffered in the war and at present it amounts to a mighty development of the production of the heavy nonferrous metals, and the complex utilization of all the valuable components of the raw material through advanced technology.

## 1. THE METALLURGY OF THE HEAVY NONFERROUS METALS IN PRE-REVOLUTIONARY RUSSIA

In 1913 33,700 tons of copper, 1,500 tons of lead, and 2,900 tons of zinc in Russia, which did not satisfy even the modest requirements of the weakly-developed industry of that time. Production of nickel did not exist at all, even though M. Danilov and G. Permyakin [1, 2, 3] attempted to organize its production from the ores of the Revda region in the Urals in the 1870 - 1890's. Every year Russia imported a considerable amount of nonferrous metals from abroad; for example, about 15 percent of the nation's total consumption of copper, 97 percent of the lead, and 73 per cent of the zinc were imported in 1913 [4]. The per capita consumption of copper in 1913 in Russia amounted to 0.2 kilograms, while it was more than 3 kilograms in England and the United States [5]. The raw materials resources of the country were not used satisfactorily and they amounted to a total of 0.6 - 4.8 per cent of the available active reserves [6].

By making use of the difficult situation of our mining industry in the period immediately after 1905, foreign financial capital, chiefly English and French, subordinated to its influence a significant part of the enterprises of nonferrous metallurgy. In 1913 80.3 per cent of the total extraction of copper in Russia was in the hands of foreign capital.

In spite of the low level of development of Russian nonferrous metallurgy from the second half of the 18th Century, progressive ideas were advanced in Russia. Thus, the use of the moist blast is now considered very important in smelting [7, 8]. Moreover, this method was applied in Russia, in the Suzunskiy Plant, in the roasting of copper matte as early as the first half of the 19th Century 9, 10, 11. It was established that when a moist blast was used, the process of oxidation of sulfides was speeded up by approximately 30 per cent and many admistures which would lower the quality of the smelted copper were removed from the gases at this time. In 1966, the engineer V. A. Semennikov was first to suggest and test the bessemerizing of copper matte in converters in the Bogoslovskiy Plant (Urals). He succeeded in smelting white matte from matte which contained 31 per cent copper, that is, to oxidize all the iron present in the matte. In 1967 V. Semennikov and N. Latetin also conducted a number of interesting experimental melts by bessemerizing with a steam-air blast in the Votkins Plant [12].

As early as the 1880 - 1890's, the casting of blocks from refuse copper slag was organized in the Bogoslovskiy Plant. The roads and areas they paved exist to this day. The production of sulfuric acid from the gases obtained in roasting copper ores was begun in this plant in 1887 [13, 14, 15].

These attempts at complex utilization of polymetallic nonferrous ores were far from being the first. Gold and silver were recovered from copper ores. The heat liberated in oxidizing sulfide ores was used for heating and smelting charges of ore. An experiment consisting of



uniting roasting and smelting in one aggregate conducted in the 1890's in the Kedabek Copper Works was interesting. Such a combined furnace, despite its small size, smelted a total of 30 tons a day, using fuel which weighed only 15.5 per cent as much as the ore [17]. Combined roasting and smelting of sulfide copper and nickel charges of ore are widely practiced at present.

The studies conducted by V. A. Vanyukov in 1904 in the Bogoslovskiy Plant determined the expediency of substituting coke for charcoal. The use of coke was started in 1906 in the Urals for smelting copper ores in shaft furnaces.

Interesting suggestions were made by the engineer Danilov (1854-1966) regarding the processing in the Revda Plant of nickel ores into nickel cast iron and by G. M. Permyakin (1878) on smelting Urals oxide nickel ores to matte with the aid of pyrites. The method of smelting cast iron from nickel ores was carried out in 1940 in accordance with the suggestion of V. V. Mikhaylov (Urals Branch of the Academy of Sciences, USSR) in the Nizhne-Salda Plant while the method of smelting nickel ores to matte has been applied since 1933 in the Ufaley Nickel Plant.

In Russian nonferrous metallurgical plants a great deal of attention was always given to the quality of the metal obtained. Thus, the method of electrolyzing copper with a copper anode was first developed in Russia. A department for electrical refining of copper was built in the "Krasnyy vyborzhets" Plant in 1900. V. A. Vanyukov conducted important studies on pyrorefining of copper in the Byysk Copper Works.

Nonferrous metallurgy in Russia fell into ruins during World War I due to the shortage of workers, the lack of timely supply of raw materials, other materials, and fuel. Copper production fell to 46.3 per cent of the 1913 level in 1917, and lead production fell to about 8 - 10 per cent of the 1913 level.

## II. THE SOVIET EPOCH. THE RESTORATION PERIOD.

The enterprises of the nonferrous metallurgical industry were nationalized by the Decree of 28 June 1918. However, the implementation of the plans outlined that year for increasing the extraction of nonferrous metals was interrupted by the Civil War and the Intervention. The nonferrous mining industry, which was concentrated in the Urals, in Siberia, and in the Transcaucasus, almost wholly turned out to be in the hands of counterrevolutionists and interventionists. When the enemy did abandon any industrial areas, under the onslaughts of the Red Army, they flooded mine shafts, dismantled the basic machinery and hauled it away from the plants and mines. Thus, they put out of commission the Kalatinskiy Plant ([See Note] The modern names of the plants will be given later.) (Now the Kirovograd Combine), the Pyshma-Klyuchevskiy and the Karabash Copper Works in the Urals, the zinc plant in Ekibastuz in the Altay, the Alaverdy Plant in the Caucasus, and others.

In the period from 1918 to 1922 no copper was smelted from ore in our country on account of the above reasons. The demand for copper was satisfied essentially by remaining stocks and scrap. These materials were reworked in plants for processing nonferrous metals in Leningrad, Kol'chugino, and Tula. According to the approximate figures of Professor V. Ye. Petrosov, the production of secondary copper in Russia amounted to 2,053 tons in 1918, 200 tons in 1919, and about 2,000 tons in 1920.

The turning point in the metallurgy of copper, as in the whole national economy, began in 1921 after the Party Congress which adopted a resolution on the transition to the New Economic Policy. Work has started on restoring the copper mines and smelters. The Korovograd Copper Smelting Combine in the Urals was the first to be put into operation in 1922. At first the smelter worked on supplies of ore which had been mined before the enterprise was stopped. A fire in the Kalstinskiy Mine, which was on the site of the smelter, disrupted the plans which provided that this mine was to be the main supplier for the smelter. It was necessary to pump out the Karpushinskiy and the Belorechensk Mines which were located about 20 kilometers from the smelter. This placed the smelter in a difficult position on account of stoppages in the supply of ore due to the poor transportation and also on account of the lack of personnel who had sufficient experience in the smelting of the ores of the Karpushinskiy Mine which were strongly mixed with zinc. V. S. Gulin [19] pointed out that the Kirovograd Smelter had lost 4,447 tons of zinc in smelting 16,538 tons of this ore in five months of 1922.

The Nizhne-Kyshtym Electrolytic Plant was put into operation in November 1922.

The reverberatory furnace, then the shaft furnace of the Pyshma Klyuchevskiy Copper Smelter (Urals) were put into operation in July 1924. Available supplies of ores in the dumps with a 3 per cent copper content and brass scrap which had come in from the outside were reworked in this smelter. The matte obtained from smelting the ore was processed in two 5-ton basic converters. In November 1925 the plant was stopped. [Sic].

Despite the presence of large stocks of red copper in the country, and the intensive reworking of brass scrap, and the reopening of the Kirovograd Copper Smelter, it was discovered as early as 1923-1924 that the copper industry could not satisfy the rapidly growing needs for this metal. The production and consumption of copper in that year are characterized by the following figures (tons) [20]:

Smelted in the Kirovograd Smelter from ores	--- 2,620
Obtained from reworking brass scrap	--- 4,340
Obtained from stocks of red copper	--- 11,420
Purchased abroad	525
Total consumed	18,905 tons of copper

The approaching "copper hunger" demanded that immediate measures be taken. The Tanalyk-Baymak Copper Smelter in Bashkiria was put into operation at the end of 1924 after a lengthy deactivation (since 1917). Rich ores from a number of small deposits with a content of 5-6 per cent copper, 12-14 grams of gold per ton, and 80-100 grams of silver per ton. The presence of up to 40 per cent fine ore compelled the use of a reverberatory furnace in addition to a shaft furnace. The matte, which contained up to 30 per cent copper, was bessemerized in acid converters. The crude copper obtained was cast into anodes and sent to the electrolytic plant for refining.

In 1925 the Alaverdy Copper Smelter was restored in the Caucasus. This smelter, which had produced about 37 per cent of all the copper smelted in the Caucasus before the war, had been deactivated from 1917 to 1925.

Even in 1921 the English capitalist L. URKVART [Transliterated, presumably Urquart], a stockholder of the Kyshtym Mining Plants Company who was led as he asserted "by strictly humanitarian considerations to aid Russia", offered to take over the Karabash Copper and the Leninogorsk Lead-Zinc Enterprises as concessions. He guaranteed that they would be put into operation and that the respective metals would be mined; in particular, he offered "out of humanity" to smelt up to 4,918 tons of copper at Karabash in 1926, even though this smelter had produced 8,754 tons of copper even in 1913. The Soviet government refused to sign the one-sided contract. Karabash was restored by our own efforts. The making of apparatus for restoration work was started in January 1925. The restoration had to be done under exceedingly difficult conditions, but the smelter started to produce copper on 22 May 1925.

The restoration of the copper smelting industry was essentially completed when the Alaverdy and the Karabash Smelters were put into operation in 1925. As a result of this work, the production of copper had increased markedly and amounted to 9,042 tons in 1924-1925 as compared with 1,785 tons in 1922-1923.

During the period of restoration of the copper industry of the USSR, Soviet scientists and engineering-technical personnel had made significant contributions to improving this branch of technology.

N. N. Barabashkin developed a technological layout for processing the slimes of the Kyshtym Copper Smelter which permitted recovery of copper, nickel, selenium, gold, and silver from them. With the participation of A. M. Gayev and N. P. Diyev, this technology was introduced in 1925 in the Kyshtym Smelter, then in the newly constructed Pyshma Smelter in 1929-1933.

Academician A. A. Baykov founded the theory of pyrite smelting [20], which was the principal method of smelting copper out of ore in the restoration period. This work established the role of coke in smelting, the conditions for obtaining rich matte were clarified along with the necessity for smelting chalcopyrite ores at a low particle size. A. A. Baykov developed a method for calculating charges for

pyrite smelting by starting with heat engineering data. He proved that oxidization of a part of the iron sulfides to ferrites in this process is essential for ensuring the required heat conditions.

A. A. Baykov's conclusions were confirmed by V. I. Smirnov [21] by analyzing the results obtained by shaft furnace smelting of chalcopyrite ores in the Kirovograd and the Karabash Smelters and later by other authors [22] who studied the behavior of charge materials in the shaft furnace smelting of sulfide copper ores.

In 1923 - 1925 V. Ya. Mostovich proved that ferrites constitute an essential component of copper matte.

By the end of the restoration period, the personnel working in the Kirovograd Copper smelter had succeeded in accumulating a considerable amount of experience in smelting zinc-containing copper sulfide ores in shaft furnaces. Mastery of smelting of charges which contained a large amount of fine particles was begun in the Tanalyk-Baymak, Alverdy, and Zangezur Smelters.

The Elektrostsinsk Plant (formerly the Alagir Plant) in the North Caucasus was the chief producer of lead and zinc in Russia. In 1913 the plant produced 97 per cent of all the lead in the country and almost all the zinc. The distillation method was used in producing zinc. The raw material consisted of lead and zinc concentrates obtained from the lead-zinc sulfide ores of the Sadon Deposit which were subjected to mechanical dressing at the Mizur Ore Concentration Plant. Losses of metals in this operation were so high that it was considered unprofitable to process ores with a zinc content of less than 16 per cent and lead less than 6 per cent in the plant. After a comparatively short period of deactivation, the enterprise was put into operation in 1922. In 1922-1923 190 tons of lead and 187 tons of zinc were produced.

In 1922 likewise, work was renewed in the Altay Combine (formerly Ridder, now the Leninogorsk Plant), whose construction had begun before the Revolution, but had not been completed. V. A. Vanyukov took an active part in restoring the Leninogorsk and the Ekibastuz Plants. Here a small ore concentration plant was opened to process quartzites which contain gold. Lead concentrates from this plant were sent to the Ekibastuz Lead Plant, in which a small shaft furnace had been built by this time. In 1922-1923, 117 tons of lead was produced in the Ekibastuz plant.

The presence of deposits of rich lead ores in Kazakhstan which lay almost at the surface of the ground and which contained no zinc or copper caused the organization and the smelting of lead in quantities up to hundreds of tons in several artisan plants in the period up to 1926.

The Elektrostsinsk Plant gradually increased its capacity. In 1924-1925 it produced 645 tons of lead, in 1925 - 1926 884 tons, and in 1926 - 1927 943 tons. The production of zinc in this plant in the same years was, respectively, 1,492, 1,888, and 2,666 tons.

During the restoration period, production men as well as scientists working in the lead-zinc industry devoted their main attention to problems of concentrating lead-zinc ores, sintering lead concentrates, smelting the agglomerate in shaft furnaces, and also to problems of the electrolytic production of zinc. In the middle of 1923 pilot facilities were built and put into operation in the Leninogorsk Plant for making electrolytic zinc. It produced the first 1.5 tons of this metal in a period from July-September 1923 [24, 25, 26]. In the Kirovograd Copper Smelter, V. A. Aglitskiy and N. P. Diyev [27] obtained the first electrolytic zinc from Karpushinskiy copper-zinc ore by means of consolidated pilot facilities.

### III. THE PERIOD OF RECONSTRUCTION AND THE CONSTRUCTION OF NEW ENTERPRISES

The decree [POSTANOVLENIE] of the government of the USSR of 10 June 1925 provided for the construction of the Krasnoural'sk Combine (formerly the Bogomolovski) which was to consist of: a copper smelter with an output of 10,000 tons a year, with future expansion to 24,000 - 25,000 tons, four mines, a broad-gauge railroad from Verkhnyaya Station to the Plant and a number of other subjects. At this time it was decided to expand the operating copper smelters and the corresponding mines and also to open new mines in the Copper-Zinc Deposits imeni Third International.

In 1925 the problem of the restoration and construction of the Karsakpay Copper Smelter in Kazakhstan was also solved. The construction of this smelter, which was intended to process ore from Dzhezkazgan, was started back in 1914. The fuel was to come from the Baykonur pits. By the time that the restoration and construction work was started, the construction was about 50 per cent complete. Most of the buildings had been built but the metallurgical furnaces and many other aggregates had not been built. The conditions for building this plant were exceedingly severe -- it was located 375 kilometers from the railroad in an uninhabited locality without water, and no skilled labor was available. In spite of all the difficulties, the plant was commissioned and produced its first copper on 19 October 1928.

In December 1925 the XIV Party Congress approved the plan for industrialization of the nation. It was necessary sharply to accelerate the tempo of development of nonferrous metallurgy. A number of measures were adopted for this purpose; in particular, a special fund for financing was set up.

Work was intensified on the expansion and reconstruction of the old enterprises which had been put into operation -- the number of operating furnaces was increased; the capacity of airblast facilities was increased; the measures were taken to return metallurgical dust to the production cycle; and work was carried on to the utilization of sulfur vapor. Thus, in the Kirovograd Smelter a belt conveyor was built for sintering metallurgical dust in 1926; the construction of a third shaft

furnace 1.4 x 11.3 meters size was completed as a supplement to the two on hand, along with changes and improvements in the removal of waste slag, the construction of a third converter with a compressor was started, and the construction of a sulfuric acid tower plant was organized to make use of the sulfur vapor from the shaft furnaces in 1927. Two SPIRLE [Transliterated] were installed in the "Elektrotsink" Plant in 1927 for roasting zinc ores [19].

As a result, the production of copper in the USSR rose to 13,606 tons in 1926-1927 against 9,042 tons in 1924-1925; the production of zinc increased in the same time to 2,666 tons against 1,492 tons.

By the beginning of the First Five-Year Period, the copper and the lead-zinc industries of the USSR had already made considerable progress. By 1928 the production of lead had reached 2,344 tons, that is, it had surpassed the 1913 level and the production of zinc (2,246 tons) and copper (1,898 tons) [Translator's note: Apparently a misprint, perhaps it should be 11,899 tons] approached it closely.

In the First Five-Year Period (1928-1932), the work on the reconstruction and construction of the enterprises connected with the principal heavy nonferrous metals had increased sharply. Capital investments in it during this period exceeded the capital outlays up to October 1928 by 7.4 times. In the First Five-Year Period the following were completed and put into operation: the Krasnoural'sk Copper Smelter (1931); the distillation zinc plants at Konstantinovka in the Ukraine (1930) and the Belovo in Western Siberia (1931); and the Sikhote-Alin' Lead Plant with hearth smelting (1932).

During the period 1929-1932, these copper ore concentration plants were put into operation, the Kirovograd, Pyshma, and Krasnoural'sk in the Urals, and Karsakpay in Kazakhstan.

Construction was started on the Chimkent Lead and the Balkhash Copper Plants in Kazakhstan, and the Chelyabinsk Zinc and the Ufaley Nickel Plants in the Urals. The production of copper in 1932 was 2.4 times greater than it was in 1927-1928 and 1.5 above the 1913 level. Lead production rose respectively by 8.1 and 13 times and zinc by 6.2 and 4.4 times. Nevertheless, the copper and the lead and zinc enterprises of the USSR could not satisfy the growing requirements and the consumption of imported nonferrous metals was significant -- 36 per cent of the nation's total consumption of copper, 60 per cent of the lead, and 56 per cent of the zinc.

During the Second Five-Year Period, the results from numerous scientific research and experimental production projects started in the USSR as early as the restoration period began to be applied in practice in the copper, lead, and zinc industries. During the years spent in putting the Kirovograd Copper Smelter and the "Elektrotsink" Plant into operation, and as a result of experimental work in the Leninogorsk Combine (1922-1926), ideas were advanced concerning the need for wider utilization of the copper-zinc and the copper-lead-zinc ores of the Caucasus, the Urals, and the Altay as complex ores. It was established that the technical-economic indices of the work of the old ore concen-



tration plants, the Mizur and the Leninogorsk, were unsatisfactory as much metal went into the tailings. It was decided, therefore, to convert to a new method of ore dressing--flotation. Many experimental and theoretical projects were carried out for this purpose (refer to the article by V. A. Glembotskiy "Concentration of the Ores of Nonferrous and Rare Metals"; page 415). The first modern plant for ore dressing by flotation of porphyry copper ores was commissioned in April 1929 at the Karsakpay Copper Smelter. About the same time the Mizur Ore Concentration Plant for concentrating lead-zinc ores (Caucasus) was reconstructed and put into operation. Later the number of ore concentration plants for processing copper ores began to increase very rapidly. At present a large portion of the copper, lead, zinc, and nickel arrive at the metallurgical plants after suitable concentration.

The commissioning of a number of ore concentration plants which produced an output rich in the principal metal and lean in admixtures with small grain size (to 0.06 millimeters) was reflected in the technology of the metallurgical enterprises. The copper smelting industry started to use roasting and reverberatory smelting; the use of hearth smelting was made possible in lead production; and it was necessary to reorganize the processes for roasting copper and zinc concentrates and sintering lead concentrates.

Thus, in the metallurgy of the heavy nonferrous metals, the study and the mastery of the following were the most important in the First Five-Year Period: the roasting of the copper concentrates in the Krasnoural'sk Copper Smelter; the roasting of zinc concentrates in multi-hearth mechanical roasting furnaces in the Konstantinovka and Belovo Plants; the sintering of lead concentrates in sintering facilities in the "Elektrotsink" Plant; the smelting of copper concentrates in reverberatory furnaces in the Krarabash, Karsakpay, and Krasnoural'sk Copper Smelters and the ore hearth smelting of lead concentrates in the Sikhote-Alin' Plant.

At the time rich lead concentrates (75 per cent lead) were processed in the "Elektrotsink" Plant. Before sintering, two fluxes were added to these ores, and before shaft furnace smelting they were thinned out by dump slag down to a 20 per cent lead content in the charge. Thus, the rich lead raw material was thinned by fluxes and slag to a quarter its original lead content and was smelted in shaft furnaces in the form of a lean charge.

As a result of successful experiments conducted directly in the plant, M. Loskutov completely replaced all the formerly used fluxes with dump slag in 1931 and enriched the charge for shaft furnace smelting up to 50 - 55 per cent lead.

As a result of this, the cost of metallurgical processing of lead concentrates was reduced and the productivity of the plant rose several times.

In 1927 planning and research work was started on the smelting nickel from the oxidized nickel ores of the Ufaley Area. Professor N. N. Baraboshkin, the engineers A. A. Mironov, B. V. Lipkin, and others

had a large share in them. The first melts of nickel ore from this area were produced in a small round shaft furnace of the Affinazhnyy Plant in Sverdlovsk, then in 1931 in a shaft furnace of rectangular cross section in the Polevskoy Copper Smelter. Subsequently the entire production layout developed for the Ufalye Nickel Plant was tested, which included: briquetting the charge, smelting the briquettes with gypsum and limestone to matte, sending a blast through it into a converter and changing it into white matte, dead roasting the latter, mixing the nickel oxide with meal and making briquettes which were then fired in retorts with charcoal.

In 1927-1929 V. A. Vanyukov suggested a method for processing the Darasun complex ores, developed a method for smelting Dzhzhkazgan copper ores to rich matte which were sent to the Karsakpay Smelter, and also participated in the study of a method of processing ores in a suspended state. During the same period he began work on establishing a complex hydroflotation method for processing pyrite cinders and ores which is of great importance in the modern copper industry.

In the second Five-Year Period (1933-1937), in accordance with the resolutions of the XVII Party Congress to "achieve particularly rapid tempos in the development and the technical re-equipping of non-ferrous metallurgy and to carry out the final conversion to a modern method for winning copper (flotation, reverberatory furnaces) and extensively to introduce the advanced electrolytic method into the production of zinc, thus winning 70 per cent of the zinc produced in 1937", the tempos of growth in the production of copper, lead, and zinc, were accelerated to a greater degree than in other branches of the national economy.

Although 1.5 billions of rubles were invested in nonferrous metallurgy in the First Five-Year Period, 4 billions of rubles were invested in the Second. Investments in the copper industry increased 5.3 times, as compared with the capital investments made in the First Five-Year Period, and 4.1 times in the lead-zinc industry.

During the Second Five-Year Period the following were put into operation: the Chimkent Lead Plant (1934), part of the Balkash Copper Smelter, the Chelysbinsk Electrolytic Zinc Plant (1935), the electrolytic department of the Elektrotsink (1934), the Ufaley Nickel Plant (1933), the Pyshma Electrolytic Copper Plant (1934), the Rezh Nickel Plant (1936), and a number of other enterprises. The commissioning of new enterprises and also a number of measures put into effect in the operating enterprises, led to a sharp increase in the output of heavy nonferrous metals; in 1935, 63,247 tons of copper were smelted in the USSR, 36,444 tons of lead, and 46,196 tons of zinc. By 1937 this led to an almost complete cessation of imports of zinc and sharply curtailed the purchase of copper and lead abroad <sup>[30]</sup>.

In 1937 the relative share (of the total production) of copper produced in reverberatory furnaces reached almost 60 per cent, and electrolytic zinc 50.9 per cent.

During the Second Five-Year Period, as in the preceding period, intensive work was done on the theory and practice of the production of the heavy nonferrous metals. V. A. Vanyukov, G. Ya. Leyzerovich, with a number of co-workers studied the roasting and smelting of copper concentrates in suspended state [31, 32]. Acting on a suggestion made by V. A. Vanyukov [33], the personnel of the Karsakpay Copper Smelter succeeded in reducing the consumption of pyrite in smelting with reverberatory furnaces, then completely ceased importing pyrites from the Urals. At the same time, the enterprises mastered the bessemerization of copper matte with a 50-60 per cent copper content [34]. Marked progress was achieved in the Krasnoural'sk Copper Smelter in roasting and reverberatory furnace smelting.

V. A. Vanyukov, D. I. Lisovskiy, V. M. Smironov, N. P. Diyev, A. A. Tseydler, and their pupils participated in the study of different separations in the Krasnoural'sk Plant [35].

Ore hearth smelting in the Sikhote-Alin' Plant was subjected to painstaking and repeated studies.

Academician G. G. Urazov developed the theory of precipitation and reaction ore hearth smelting of lead ores and concentrates. N. A. Gosteyev [36] and F. M. Loskutov [37] studies the effects of different factors on ore hearth smelting. In particular, the necessity was proved for caking the fine dust formed in the work of the hearths. Comparative studies made at the Leninogorsk Lead Plant showed that the agglomerate obtained from conveyor sintering machines was reduced to metal more rapidly than that obtained in agglomerating boilers on account of less fusion. Experience with sintering of lead ores and concentrates on a conveyor machine in the Chimkent Plant showed that losses of lead due to volatilization were caused to a large extent by the amount of blast used during sintering and the dampness of the charge. During the same period lengthy studies were started in the Leninogorsk Lead Plant on the conditions for smelting agglomerate containing much zinc and copper simultaneously in shaft furnaces. The difficulty was due to the fact that it was necessary to leave a large amount of sulfur in the charge in order to eliminate the harmful effect of the copper on the smelting, and to liquidate the harmful effect of the zinc, on the other hand, it was necessary to remove the sulfur during sintering. Smelting copper-zinc materials with pyrite in a special shaft furnace to matte and contaminated crude lead was also organized in this plant [38].

Intensive studies were carried out during those years on problems of producing zinc by electrolysis: roasting zinc concentrates, leaching out [39], and especially the purification of solutions, and the process of electrolysis itself. In 1934, V. G. Ageyenko, and Ye. A. Tsaginkan [40] studies the elimination of cobalt from zinc solutions. In 1936-1937 the production of cadmium from copper-cadmium cakes obtained as a waste product of zinc production in the Leninogorsk and the Elektrotsink Plants [41] was mastered.

A great amount of experimental and research work was done in the Second Five-Year Period on the metallurgy of nickel, for example, on the formation of ferronickel during smelting of oxide ores in a shaft furnace and on the smelting of materials rich in magnesium oxide in the same aggregate [42]. As a result of the studies, the Ufaley Plant was converted in 1935 to producing metallic nickel from its oxide in electric arc furnaces. P. S. Kusakin (Urals Branch of the Academy of Sciences, USSR) had a large part in this work.

In accordance with the resolutions of the XVIII Party Congress the task was set for the Third Five-Year Period to increase the production of nonferrous metals to proportions which would ensure satisfaction to the rapidly growing requirements of the national economy and the defense of the nation. The production of copper in 1942 was to be increased 2.8 times. In this connection it was necessary to complete the construction of the Balkhash, Mednogorsk, and the Sredneuralsk Copper Smelter Combines, also the Severonikel' Plant, and the Yuzhuralnikel' Plant, at the same time improving production in existing plants. It was especially essential to intensify work on the complex utilization of raw materials.

In 1938 the Yuzhuralnikel' (South Urals Nickel Combine) was commissioned, in 1939 the Mednogorsk Copper-Sulfur Plant which produced elementary sulfur and crude copper, also the very large flotation plant in Leningorsk. In 1939-1940 the majority of the departments of the Severonikel' [Abbreviation not known, apparently Northern Nickel] Plant and the Sredneuralsk Copper Smelter had started work.

In the Third Five-Year Period the main attention in nickel metallurgy was devoted to mastering the smelting of oxidized nickel ores. The smelting of sulfide ores and agglomerates in shaft furnaces was also mastered in the Ufaley, Rezh, Yuzhnouralsk Plants, and in the Severonikel' Plant.

During these years the production of cobalt from converter slags of nickel production was begun; the efforts of scientists and production personnel led to rapid mastery of this new product. S. A. Pletnev [43] and A. A. Tseydler were occupied with developing a rational procedure for extracting cobalt from Soviet raw materials. V. I. Smirnov and D. I. Kosovskiy studied the conditions of work of shaft furnaces for smelting converter slags. N. Demenev, N. P. Diyev, P. S. Kusakin, and I. S. Beresnov studied the separation of cobalt into matte by processing liquid converter slags. I. G. Shcherbakov, M. A. Loskarev, and A. G. Loshkarev developed a method for extracting cobalt from metallic nickel and white matte by a procedure which included electrolysis.

V. A. Vanyukov, F. M. Luskutov, A. A. Tseydler, and others conducted a whole series of studies of the properties of slags from copper, nickel, lead, and tin melts with the objective of reducing irreversible losses of metals.

The center of attention in the field of copper metallurgy was improvement of the thermal performance of reverberatory furnaces [44, 45, 46] and their design, also perfecting the technology of this type of

smelting [47]. Important studies were made in the electric refining of copper and the reworking of semi-finished products from this branch of production [48, 49, 50]. In the metallurgy of lead and zinc, attention was devoted to sintering, shaft furnace smelting of lead agglomerate, and particularly to dust catching and the complex utilization of the dust [5], [52]. Since the hydrometallurgical production of zinc was comparatively new in the USSR, a great amount of scientific and practical production work was devoted to it during the Third Five-Year Period.

#### IV. THE WORLD WAR II PERIOD

The development of war industries sharply increased the demand for nonferrous metals. On the other hand, the production capacity of nonferrous metallurgy declined in the first part of the war. The zinc plant in Konstantinovka, the Elektrotsink Lead-Zinc Combine, and Ser-veronikel' were put out of commission. The Noril'sk Mining and Metallurgical Combine experienced a great deal of difficulty.

A large amount of new construction was developed under severe conditions in the rear eastern regions of the country. Work proceeded on the reconstruction and expansion of nonferrous metallurgical plants of the Urals, Siberia, and Kazakhstan with use of evacuated equipment. Thus, a department for the electrolysis of nickel was put into operation in the Yuzhnoural'sk Nickel Plant during these years.

Important work was done by the Commission for Mobilization for the Resources of the Urals, Western Siberia, and Kazakhstan for Defense Needs which was established under the Academy of Sciences, USSR. The Scientists and engineers in the Commission cooperated with production workers to search for ways to increase the output, to improve its quality, and to improve the indices of production. Much was done on problems of ferrous and nonferrous metallurgy by the Deputy Chairman of this Commission, Academician I. P. Bardin. In the Commission, Academicians A. A. Skochinskiy and L. D. Shchevyakov, Associate Members A. A. Ivanov, I. N. Plaksin, D. M. Chizhikov, P. F. Antipin, and also V. I. Smirnov, K. M. Charkviani, N. P. Diyeu, V. V. Rikman, and others worked directly on problems of nonferrous metallurgy. In conjunction with plant personnel, the Commission studies and presented to the government a number of suggestions on the Kirovograd, Mednogorsk, Karsakpay, Balkhash, and Irtysh Mining and Metallurgical Copper Combines, on the Ufalye and the Yuzhnoural'sk Nickel Combines, on the Chelyabinsk and Belyaev [sic] Zinc Plants, on the Leninogorsk Lead-Zinc Combine, and on a number of other large enterprises of nonferrous metallurgy in the eastern part of the USSR. Special attention was devoted at this time to the complex utilization of raw materials: to the use of metallurgical gases to increase the production of sulfur and sulfuric acid; to the recovery from copper-zinc, copper-lead-zinc, nickel-cobalt ores of their valuable constituents--copper, lead, zinc, nickel, cobalt, cadmium, antimony, selenium, tellurium, and others.



On the basis of an analysis of the activities, for example, of the Leninogorsk Lead-Zinc Combine, recommendations were made on separating copper sulfides from the bulk of lead-copper concentrate into a special fraction, thus reducing the copper content of lead concentrates down to 2 per cent. It was also recommended that the smelting of materials under processing which were rich in copper should be smelted in a special furnace. Compiling a cadmium balance for the entire output of the Leninogorsk Combine permitted the finding of sectors of greatest losses of this valuable metal and suggesting suitable measures for decreasing them. In particular, certain changes in the roasting and leaching of cadmium raw material helped to decrease the cadmium content of lead cakes.

On the basis of work done by D. M. Chizhikov, and N. N. Sevryukov [52] at the Belovo Plant a method was put into practice of scrubbing vaporized zinc in vertical carborundum columns. Very important research and industrial projects were organized to separate cobalt from sulfide copper-cobalt and oxidized nickel ores of the Urals and also for utilization of cobalt ores. Success was gained in obtaining pyrite cobalt concentrate by selective floatation. The use of a number of improvements in the reworking of cobalt-containing converter slags markedly increased its recovery from this raw material into matte. The recovery of cobalt in final production was increased as a result of improvements in the chlorinating roasting of matte, also by developing the conditions for electrolytic solution of matte and white matte enriched with cobalt and making high-grade anodes of these materials.

Research done on studying the optimal slags when smelting copper, nickel, lead, and cobalt lead to increasing the recovery of these metals in smelting the basic raw material. In the Yuzhnoural'sk Nickel Plant, for example, the nickel content of the dump slags amounted to 0.26 per cent in 1939, but was reduced to 0.22 per cent in 1944. In the Leninogorsk Lead Plant when smelting crude lead in shaft furnaces, measures were taken to obtain stable slags (up to the time these measures were taken, their composition varied within very wide limits: 17--29 per cent for  $\text{SiO}_2$ , 34 - 41 per cent for  $\text{FeO}$ , 6 - 12 per cent for  $\text{CaO}$ , 12 - 17 per cent for  $\text{ZnO}$ ). The composition of optimal slags for cobalt smelting was studied by D. I. Lisovskiy.

Improvement of the organization of catching metallurgical dust had a marked effect. On the basis of work done by the Urals branch of the Academy of Sciences, USSR and the Ufaley Nickel Plant [54], dust catching from the converter department of this plant was organized. Additional sections of beta filters, et cetera, were put into operation in the Leninogorsk Plant.

During World War II, special attention was devoted to the metallurgy of nickel and cobalt, which were of great importance in military technology. Study of reducing sulfidizing smelting of nickel agglomerate in shaft furnaces in the Yuzhural'sk Combine and experiments on its intensification carried out in 1942-1943 yielded markedly improved performance. A sharp increase in the amount of air forced into the furnace



and proper loading of materials in the charges resulted in increased smelting of the agglomerate and in decreased coke consumption. An number of similar measures lead to a marked increase in the production of nickel and cobalt during World War II.

## V. THE PERIOD FOLLOWING WORLD WAR II

The Fourth Five-Year Plan (1946-1950) provided for the significant development of the electrification of the country, machine building, and increasing the capacity of ferrous metallurgy and the chemical industry. Increases in the capacity of active mines, ore concentration plants, and metallurgical plants, also the construction of new enterprises were specified.

First place in the program of capital construction for the copper industry was assigned to building mines and ore concentration plants for the Dzhezkazgan Combine. Construction of a new copper combine was also started in Bashkiria. Construction was started on a copper-molybdenum combine on the base of the Kadzhar Deposit in the Armenian SSR.

A great amount of work was done in the Fourth Five-Year Period on the further development of the lead-zinc industry: the second stage of the Tekeli Combine was put into operation; the ore base of the Chirkent Lead Plant was strengthened; new sections of the Leninogorsk Ore Concentration Plant and the Zyryznovsk Mine were put into operation; new mining and ore concentration enterprises were built at the Bere-zovskiy and the Belousovskiy Deposits; the Leninogorsk Lead Plant was expanded; the construction of the Ust'-Kamenogorsk Lead and Zinc Plants was started. The capacity of the Chelyabinsk and Belovo Zinc Plants was increased.

Great changes were made in the nickel industry: the Pechenga Nickel PECHENGANIKEL' Combine was put into operation (1946), restoration of the Severonikel' Combine was completed, the capacity of the Noril'sk Combine was increased. Certain improvements were made in the Yuzhnoural'sk and the Ufaley Nickel Plants.

In the Fourth Five-Year Period, 12 billion rubles were appropriated for developing the nonferrous metallurgy of the USSR as a whole, that is, 2 billion rubles more than for all the prewar Five-Year Periods (1928-1940).

The problems of the complex utilization of ores, concentrates, and fluxes were studied still more intensively in this Five-Year Period than in the preceding ones. Increasing the scale of production required the use of leaner ores, for example, lead ores with 1 per cent lead content, and copper ores with still lower copper content. The unceasing work on perfecting methods of flotation was of great importance in solving this problem.

A lengthy study of the processing of polymetallic ores and concentrates by sulfating roasting suggested by A. Ye. Makovetskiy [55] in 1923 showed that it could be used expediently in industry, especially in the processing of the intermediate products of the Leninogorsk Ore

Concentration Plant. Some valuable material from this study was also used by enterprises which operated according to a standard procedure. In this method, which consists of processing the complex polymetallic sulfide raw material with 50 per cent sulfuric acid at 155 - 170 degrees, 96 per cent of the zinc, more than 95 per cent of the copper, 98.6 per cent of the silver go into the solution and 99.7 per cent of the gold is concentrated in the copper-lead cake. Over 50 per cent of the sulfur in the raw material goes into commercial production.

Large scale studies on the electrolytic refining of metallic nickel and copper [56, 57] obtained from sulfide copper-nickel ores, also on methods for purifying solutions from admixtures opened new possibilities for improving the quality of cathode metal and outlined a way for increasing the recovery of cobalt and palladium from the raw material.

Application of the method of flotation separation of copper-nickel matte [Maslyanitskiy] to copper and nickel sulfide concentrates simplified the processing copper-nickel ores and led to an increase in the recovery of these and accessory elements in the finished product.

The GIPRONIKEL' Institute worked out the conditions for obtaining a method for the electrolysis of high-grade nickel and cobalt. The GINTSVETMET Institute suggested new methods for extracting selenium from the slimes of copper electrolysis.

Experiments were conducted in the Belovo Zinc Distillation Plant in 1948-1949 on distilling cadmium and lead in sintering zinc materials [58]. A certain amount of table salt was added to the charge for this purpose and the distillation of cadmium rose to 80 - 85 per cent. However, this method has not as yet been applied.

In view of the importance of the problem of complex utilization of ores, concentrates, and fluxes in nonferrous metallurgy, the Ministry of Nonferrous Metallurgy of the USSR called an all-union conference in Sverdlovsk in 1947. This conference took up the questions of the development of nonferrous metallurgy in the new five-year period and the problem of raw materials and their complex utilization not only in the production of the basic nonferrous metals (and sulfur), but also the rare, still little-used elements.

In order to increase the degree of complex utilization of raw material and to raise the productivity of aggregates in the Fourth Five-Year Period, a number of large-scale laboratory and plant projects were conducted on the use of air enriched with oxygen in the copper smelting, nickel, lead, and zinc industries [59, 60].

Important projects on the use of oxygen during shaft furnace smelting of oxidized nickel ores were carried out by the Yuzhural'nikel Combine with the aid of the GINTSVETMET and GIPRONIKEL' Institutes.

V. I. Smirnov, M. A. Chernyak, and M. A. Abdeyev [61] proved that even 24-25 per cent enrichment of the air blast with oxygen in shaft furnace smelting of lead results to a 20 - 30 per cent increase in the amount of charge processed and to a 15 - 20 per cent reduction in the losses of lead and noble metals in slag and dust.

The projects of the UNIPROMED' [Abbreviation not known, apparently Urals Scientific Research Institute for the Copper Industry] (A. A. Babadzhan) and the Kirovograd Plant on mastery of the method of pyroselection and of the Moscow Institute of Nonferrous Metals and Goldsmiths M. I. Kalinin (A. V. Vanyukov, N. I. Utkin) on development of a method for smelting in a liquid bath and on the study of cyclone smelting.

Study of the thermal engineering of reverberatory furnaces and increasing their heat capacity [62, 63], improving the form of the flame, improving the preparation of furnace charges and methods for leading them led to a sharp improvement in the productivity of reverberatory smelting and a reduction in losses of metals in the slag. A decrease in fuel consumption and achievement of higher technical-economic indices in this sort of smelting made possible improvements in design which are introduced in reverberatory furnaces by the Balkhash and the Krasnoural'sk Copper Smelters and the GIPROTsvETMET and UNIPROMED' Planning Institutes.

The theoretical work of Academicians A. A. Baykov, and G. G. Uruzov, and of the Honored Scientists and Engineers V. A. Vanyukov, and N. P. Aseyev, V. Ya. Mostovich, A. N. Vol'skiy and Kh. K. Avetisyan exerted a great influence on the metallurgy of the heavy nonferrous metals.

In 1950 the production of copper surpassed the 1945 level by 82 per cent, the production of zinc and lead, respectively by 2.33 and 2.39 times. The production of rare elements contained in the raw material used in copper and lead-zinc enterprises markedly surpassed the prewar level. However, the results obtained still did not satisfy the growing needs of the country. Therefore, the directives of the XIX Congress of the KPSS in respect to the Fifth Five-Year Plan again set more rapid tempos increasing the production of nonferrous metals as compared with the tempos of growth set for the output of ferrous metals. With a general increase in industrial production of 70 per cent in the Fifth Five-Year Period, the production of copper was to be increased by 90 per cent, nickel, by 1.53 times, lead by 2.7 times, and zinc by 2.5 times. In accordance with this, it was provided that large capital investments were to be made in this branch of industry. Special attention was devoted in the plan for mechanization of laborious processes, automation, intensification of production processes, the use of oxygen blasts, increased complex recovery of metals from ores, the ancillary use of sulfur vapor from polymetallic ores and assurance of further growth of the output of high-grade metals.

In the Sixth Five-Year Period, as in the Fifth, attention was directed mainly to introducing methods of complex processing of ores and concentrates. For this purpose, they began to make use of combined procedures in production work for processing raw materials, including pyrometallurgical and hydrometallurgical methods. At the same time, however, these methods underwent important changes.

For example, the Moscow Institute of Nonferrous Metals and Gold imeni M. I. Kalinin, under the guidance of A. V. Vanyukov, successfully conducted projects jointly with the Yuzhural'nikel, Combine on concentrating oxidized nickel ores after they had undergone preliminary reduction-sulfidizing roasting.

In order to raise the degree of complexity of the utilization of raw materials, the extensive introduction of zone roasting of sulfides and oxides was begun. The development of basic theoretical data for these processes was started in the Institute of Metallurgy of the Urals Branch of the Academy of Sciences, USSR, and in the GINTSVETMET Institute. There they studied, over a wide range of temperatures and other conditions, the mechanics and kinetics of the oxidation of different sulfides, and also the reduction of oxides. The technological conditions for zone roasting were worked out in the GINTSVETMET Institute by G. Ya. Layzerovich and in the Elektrotsink Plant by G. M. Shteyngart [64]. In addition to increasing the productivity of the roasting furnaces in zinc plants, a 2-3 per cent increase was achieved in the solubility of zinc in the electrolyte which was developed. Increasing the sulfurous anhydride to 8 - 10 per cent (against 4-5 per cent in the usual roasting) led to an increase in the capacity of sulfuric acid plants of 30 - 40 per cent. All the zinc plants are now being converted to this method of roasting. Its use is also being started in the Sredneural'sk Copper Smelter.

Increasing the lead content in its concentrates (which also contained zinc and copper) required changes in the methods for processing them as it had been necessary to make the charges much leaner (approximately down to 30 percent lead) in the usually accepted procedure and to lose a great deal of zinc. The use of electric furnaces for this purpose with a bath of carbon blocks made it possible to distill off the zinc, to raise the recovery of lead in the form of crude metal by 3- 5 per cent, and to improve the working conditions of the attending personnel. Exploitation of the electrothermal facilities built for this purpose in the Irtysh Combine in 1953 - 1955 confirmed these conclusions [65].

During the last 5- 6 years facilities have been built in the lead smelting plants of the Ust' - Kamenogorsk and the Leninogorsk Combines for fuming dump slags. After fuming they can be used in place of fluxes in shaft furnace smelting of lead. The study of fuming processes conducted by A. I. Okunev in the UNIPROMED' Institute [66, 67, 68] marked an improvement of the fuming process in lead plants and in the conditions for processing the zinc slags of copper smelting enterprises.

VELTSEVANIE [Translator's note: English term not known; this is a continuous process in which the raw material is fed into an inclined rotary furnace in which a blast of air is introduced at the lower end, counter to the movement of the material, which must remain friable during the process. The volatile zinc and lead oxides are passed through filters for recovery.] of the cake from electrolytic zinc plants is widely used. In this processing, the recovery of zinc reached 91 - 99%

per cent and lead 92 per cent. In the Belovo Zinc Plant the oxides are granulated and sent to distillation retorts.

Extensive studies were made during the Fifth Five-Year Period on dust catching and the complex processing of metallurgical dust. Workers from the metallurgical and chemical institutes of the Urals Branch of the Academy of Sciences, USSR, the Kazakh and the Ukrainian Academies of Sciences, the GINTSVETMET and UNIPROMED' Institutes, and also enterprises of the copper, nickel, and lead-zinc industries participated in the work. The projected procedures for processing these materials passed the stage of plant tests.

Every year sees ever wider introduction, in the first instance in the nickel industry, of electric smelting of concentrates to matte in arc furnaces. The application of this method to the smelting of copper sulfide concentrates, too, is being examined. The conducting of discussions [69, 70] has permitted many interesting decisions; in particular, suggestions have been made in regard to increasing the recovery of copper in matte, zinc in slag, and a number of elements from the ore in the vapor phase.

In accordance with the directives of the XIX Congress of the KPSS, large projects were undertaken with the purpose of expanding the use of oxygen in the production of the heavy nonferrous metals. An oxygen-making station with a capacity of 1000 cubic meters of oxygen per hour was built in the Yuzhnoural'sk Nickel Plant. Experience with the shaft furnace smelting of nickel agglomerate with an air blast containing up to 25 per cent oxygen showed that the productivity of the plant increased 25 - 30 per cent, the coke consumption was reduced 15 - 20 per cent, and the recovery of nickel in the matte was increased 3 - 5 per cent, chiefly due to decreased losses of nickel in the dust [71]. Work is being continued on the use of oxygen in this plant. During the Fifth Five-Year Period, a number of melts were made in a 40-ton converter operating on an air blast containing 33 - 35 per cent oxygen in the Krasnoural'sk Copper Smelter.

In 1956 - 1957 a number of oxygen stations with a capacity of 3600 cubic meters of oxygen per hour were built for the nonferrous metallurgical plants of Kazakhstan.

During the Fifth-Five-Year Period vacuum refining was used in the Chimkent Plant for separating lead from zinc and vacuum refining was used in the Chimkent silver scum [72, 73]. The refining of lead from bismuth [sic] was introduced in all lead plants. [Translator's note: Perhaps this means removing traces of bismuth from lead ores]. Studies conducted by G. G. Urazov and his co-workers V. S. Lovchinkov and B. M. Lipshits established a scientific and practical basis for the alkali method for refining lead, introduction of which method in the Chimkent and the Ust'-Kamenogorsk Lead Plants improved the technological indices of the process for pyrrifying lead by removing arsenic, tin, and antimony.

The technology of electrolytic zinc plants was markedly changed. For instance, wider use was made, in the process of leaching, of oxidizing ferrous iron to ferric iron with the aid of air, which partially replaced manganese dioxide. Study of this problem on a laboratory scale and on a plant scale showed that the presence of manganese in the electrolyte caused a number of difficulties in the processes of thickening and electrolysis [74]. The purification of the electrolyte by removing cobalt with ethyl zanthate is widely practiced. The current density in this electrolysis was increased. All this led to the fact that the Fifth Five-Year Plan for the total amount of industrial production was fulfilled by 1 May 1955.

In many copper, lead, nickel, and zinc enterprises, the indices [figures on] of the recovery of metals and the utilization of equipment (specific productivity) were notably improved. The complex utilization of ores was improved: sulfur vapor, cadmium, selenium, bismuth, and a number of rare elements were recovered as by-products.

The recovery of copper in the copper smelters increased a little during the Fifth Five-Year Period, the recovery of lead as crude metal increased about 2 per cent, the recovery of zinc by 5 - 15 per cent, and the recovery of nickel by 1 - 7 per cent.

During the Fifth Five-Year Period there was a significant reduction in the production costs of metallurgical separation, calculated on the basis of one ton of metal. For example, the cost of copper in the Krasnoural'sk Plant was reduced from 871 rubles in 1950 to 721 rubles in 1955, lead to in the Chimgent Plant during those years from 1070 to 898 rubles, and nickel in the Severonikel' Plant from 13,287 to 8,862 rubles.

In the Sixth Five-Year Period, the following are to be introduced into production: the multi-stage dressing of ores, the combined method for concentrating and hydrometallurgical processing of ores, zone roasting, and electrothermal and autoclave processes. A great deal of attention should be devoted to scientific research and experimental work on searching out more economical processes for winning nonferrous metals, and also on further improvement and a mastery of the technology of production of rare metals and comprehensive study of their properties and fields of application.

The reconstruction of the Sredneuralsk Plant has been going on since 1956. In this plant, also in the Balkhash Copper Smelter, experimental cyclone smelting has been conducted in reverberatory furnaces in accordance with a method suggested by the Kazakh Academy of Sciences and the All-Union Institute of Fuel Utilization in Sverdlovsk. The Urals Branch of the Academy of Sciences, USSR and the UNIPROMED' Institute are studying a process for oxidizing copper sulfide charges in a flame burning in oxygen. The building of slag distillation facilities is planned for the Krasnoural'sk Plant and construction had been started on facilities for processing metallurgical dust.



A sulfuric acid department is being built in the Mednogorsk Copper-Sulfur Plant, facilities will be started in the Kirovograd Copper Smelter for utilizing lean sulfur gases, the departments for processing dust and distillates will be re-equipped in the Ust'-Kamenogorsk Lead-Zinc Combine and, in particular, a method will be used for sulfating lead dust to win cadmium, indium, and thallium. Here, too, slag distilling facilities have been built and also facilities for freeing lead from zinc by vacuum smelting.

In 1955-1957 the floatation separation of copper-nickel white matte was started in the Severonikel' Plant.

The conditions for winning pure nickel through formation of its carbonyl [Translator's note: Apparently the Mond process] were studied. In the near future the carbonyl method will be used in the Severonikel' Plant and in Krasnoyarsk.

Intensification of the processing of lead, zinc, and copper concentrates through the use of oxygen is planned for the Ust'-Kamenogorsk Balkhash, and Irtys' Plants and in the Yuzhuralnikel' Plant.

All these processes require painstaking study in the scientific research institutes.

#### RESEARCH ON THE METALLURGY OF TITANIUM

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In spite of the fact that metallic titanium was obtained for the first time in 1925 [3], that is, more than 100 years ago, there exist incorrect notions of metallic titanium as a brittle metal which is difficult to work. Titanium attracted almost no serious attention on the part of research workers in any country in the world. Only 40 years ago just 5000 tons of ilmenite concentrate were produced by the entire world.

In the last few years, however, a colossal development has been seen in the metallurgy of titanium, as evidenced by an increase of about 250 times in the extraction of ilmenite concentrate, and titanium itself has been transformed from a laboratory rarity into a structural metal which is being produced in the tens of thousands of tons each year. Titanium dioxide has found wide use in the pigment industry and other branches, and at present is being produced at the rate of over 600,000 tons a year.

New applications are found every day for this metal which possesses splendid properties.

The strength of titanium and its alloys is twice that of iron and six times that of aluminum. This strength is retained up to 570 and even up to 650 degrees. Titanium possesses high resistance to corrosion both under ordinary atmospheric conditions and in sea water. Titanium surpasses stainless steel and nickel base alloys in respect to anticorrosion stability and its low specific gravity permits a reduction in machine parts down to 40 per cent of those parts made of stainless and other grades of steel. Titanium and its alloys can find wide application in aviation, shipbuilding, in the chemical, electric power, medical, and other branches of machine building and instrument building industries. Titanium has become an important strategic material which is finding ever wider use in different branches of engineering. Titanium has, so to speak, lived through its rebirth. There can be no doubt that in the near future titanium will become one of the most widely used metals in engineering, along with iron, aluminum, and magnesium. This is all the more case because it is widely found in nature (0.63 per cent of the weight of the earth's crust), occupying tenth place after oxygen, silicon, aluminum, iron, calcium, sodium, potassium, magnesium, and hydrogen.

The development of modern technology -- mastery of obtaining pure inert gases in adequate quantities, development of vacuum techniques, particularly smelting metals in vacuum and other achievements of science and engineering have created the essential conditions for the industrial winning of titanium.

At present the metallurgy of titanium is a new branch of production which includes a number of technological problems such as: the production and processing of titanium slags and concentrates to titanium tetrachloride and titanium dioxide; the metallothermic reduction of titanium compounds; the processing (vacuum separation) of reaction masses; and the smelting of metallic titanium. However, the metallurgy of titanium has not been sufficiently studied as yet nor has it been mastered in an industrial sense, which is a consequence of the little interest in titanium in the past.

In Russia serious scientific research in the field of the geology, chemistry, and metallurgy of titanium were begun only in 1915. This research was established on the initiative of A. Ye. Fersman of the Raw Materials Commission; and such well known scientists as Academician D. S. Belyankin, M. S. Maksimenko, and others participated in the work. "One of the first basic problems advanced by the chairman of the Commission, A. Ye. Fersman, from the very beginning of its existence, was the problem of titanium, both in view of the real necessity of obtaining titanium tetrachloride for a whole series of applications in military technology and in view of its importance in the metallurgical and other branches of industry" [1].

The Commission did important work on discovering reserves and developing industrial methods for obtaining the tetrachloride, the dioxide, the carbide, and other titanium compounds.

In 1917 an industrial method was developed for obtaining  $TiCl_4$  from ilmenite (with a transition through titanium carbide).

The Commission also did important work on finding uses for titanium in technology, for which purpose questionnaires were sent out to plants and organizations concerning the use of titanium ores in Russia.

According to data from the questionnaires, the engineer Khrushchev was working in a machine building plant in Orel' at that time on obtaining titanium alloys by a thermite method.

# I. RESEARCH IN THE FIELD OF PRODUCTION OF TITANIUM SLAGS

Rutile concentrates and slags rich in titanium which are obtained from smelting ilmenite and titanomagnetite concentrates are used in the production of metallic titanium.

As early as 1917 the Titanium Commission pointed out the prospects of titanium slags as a raw material for winning titanium. The commission considered that smelting ilmenite with an addition of calcium oxide to the charge as a flux, followed with chlorination of the titanium slag freed from iron and its oxides would be one of the most profitable methods for obtaining titanium tetrachloride [1].

A comparison of modern prices of different types of raw material (Table 1) shows that ilmenite and slags from the smelting of ilmenite concentrates are the cheapest raw material. However, the metallurgy of titanium should be directed toward the use of titanium slags as the raw material, since the problem of use of titanium raw material with recovery of titanium and iron can be solved in a complex way in this case [2, 4, 5].

TABLE 1

Type of Raw Material	Price per Milogram of Raw Material in Dollars	Price per Kilogram of Titanium in the Raw Material
Ilmenite concentrate ( $TiO_2$ content 55 per cent)	0.02	0.06
Rutile concentrate ( $TiO_2$ content 92 per cent)	0.13	0.23
Titanium slag ( $TiO_2$ content over 70 per cent)	0.040	0.095
Titanium dioxide pigment	0.53	0.88
Titanium tetrachloride	0.88	3.46

At present, rutile concentrates [2, 4] and, to some extent, titanium slags from smelting ilmenite ores and concentrates containing up to 80 per cent  $TiO_2$  and titanomagnetites (up to 50 per cent  $TiO_2$ ) [5, 6] are the basic raw materials for producing titanium tetrachloride.

The prospects are that rutile concentrates can only partially satisfy the needs of titanium metallurgy inasmuch as their extraction does not exceed 42,000 tons a year and further expansion is limited. The annual output of ilmenite concentrate amounts to more than 1,100,000 tons, and the possibilities for expanding that production are actually not limited [7]. Consequently, the titanium industry of the future should be linked only with a mass sort of raw material like ilmenite concentrate.

As regards titanomagnetite concentrates, the problem of their use in producing titanium tetrachloride has not been studied sufficiently as yet. However, the low cost of titanium slags obtained from smelting titanomagnetites (they cost only a fraction as much as the slags obtained from smelting ilmenite concentrates) and the significant concentration of titanomagnetites in primary deposits with colossal reserves [8], will permit the construction of large metallurgical combines based on them with complex utilization of the iron, titanium, vanadium, niobium, and other elements contained in these ores.

At present, ilmenite concentrates should be considered the principal raw material for the production of titanium in the Soviet Union.

More complete separation of titanium and iron is the most important problem in the use of this type of material.

Their reduction smelting is considerably more effective than direct chlorination of ilmenite concentrate [6] or sulfuric acid decomposition [9], and solves this problem more simply, resulting in winning commercial cast iron and titanium slag.

the problem of obtaining high-titanium slags which meet the requirements for their chlorination has been essentially solved in the USSR. This technology is different from the process developed in the plant at SOREL [Translator's note: Apparently the town, of Sorel, Canada, located near Montreal] [6] and the process developed in Japan which is based, according to the literature, on obtaining slags with a titanium sesquioxide content of more than 50 per cent [5].

The process for obtaining titanium slags developed in the USSR permits obtaining slags with a minimal iron content (in terms of the lower oxide, up to 4 per cent), with a low smelting temperature and a small expenditure of electric power (markedly lower than 3000 kilowatt-hours per ton of slag).

The first melts of titanium ores in Russia were made by V. N. Lipin in 1897 with a charcoal-fired blast furnace of the Vidlitskiy Plant (in the former Olonetskaya Guberniya) [1, 10]. Systematic studies on the smelting of titanomagnetites were started in 1929. Seven plant runs from blast furnaces and numerous laboratory studies were made in a decade [11, 12, 13, 14, 15, 16, 17].

The research in the field of smelting titanomagnesites [translator's note: Presumably a misprint in the original text, apparently should be titanomagnetites] was headed by Academician E. V. Britske. His pupils, K. Kh. Tagirov, and I. V. Shmanenkov were the first to develop the technology of continuous blast furnace smelting of lump titanomagnetites with alkali added to the charge, thus obtaining titanium slags with increased titanium dioxide content [14, 15, 16].

Academician A. M. Pavlov made an important contribution to the study of blast furnace smelting of titanomagnetites.

Studies of the processes of reduction of titanomagnetites carried out under the leadership of K. Kh. Tagirov and V. B. Mikhaylov showed that ilmenite is the part of the ore which is difficult to reduce. It is reduced directly only in the lower layers or even in the hearth of the blast furnace. The main difficulties in the smelting of high-titanium charges are caused by the fusibility of the ilmenite (1380 degrees) and the early development of the processes of primary slag formation. Primary titanium slags are unstable (they crystallize in a very small range of temperatures) and the fluctuations of temperature which can exist in a blast furnace on the slag-forming layer are certainly sufficient to cause these slags to pass from the molten state to the solid, thus causing the formation of crusts and creating difficulties for even descent of the charge.

It should be noted that titanium slags under reduction conditions are not in equilibrium and if crystallized at a low temperature can be remelted at a higher temperature or, if the reducing processes have proceeded further with the formation of  $Ti_2O_3$  and  $Ti(O)C$ , they cannot be melted at all under blast furnace conditions.

In order to ensure a continuous smelting process, it is essential to introduce a considerable amount of flux into the charge to stabilize the primary blast furnace slags. This lowers the titanium dioxide content to such a low level (20 - 25 per cent) that processing these slags for titanium dioxide becomes unprofitable.

Thus, smelting titanium ores in a blast furnace did not yield favorable results. Obtaining slags with a high titanium dioxide content became possible in low shaft electric arc furnaces where the processes of primary slag formation would have such a decisive influence as contrasted with smelting in blast furnaces.

In 1949-1950 the Academy of Sciences and the MChM [Abbreviation not known, apparently the Ministry of Ferrous Metallurgy], USSR smelted titanomagnetites in the Kuznetsk Metallurgical Combine, thus obtaining high-titanium slags and vanadium cast iron. Agglomerates were smelted with a continuous process with closed top [of the furnace]. At the same time, the expenditure of electric power amounted to 2200-2400 kilowatt-hours per ton of cast iron. As a result of these projects, the technology of processing titanomagnetites by electric smelting was developed. The composition of the slag obtained corresponded to the slags obtained from smelting titanomagnetite sands in Japan which were used for obtaining titanium tetrachloride.

The smelting done in the Kuznetsk Metallurgical Combine showed that the productivity of the furnace was determined by the speed at which the iron oxides were reduced; therefore intensification of these processes can markedly increase the technical indices of smelting. The results of the smelting at the KMK [Kuznetskiy metallurgicheskiy kombinat - Kuznetsk Metallurgical Combine] should be regarded as minimal.

Research conducted by the metallurgical institutes IMET AN SSSR [Abbreviation not known, apparently Institute of Metallurgy of the Academy of Sciences, USSR] and the reducing iron oxides could be achieved by special preparation of the materials in the charge (joint briquetting of the mixture of the oxides to be reduced and the carbon reducing agent) and separation of the processes of reduction and slag formation in different facilities. Good results were obtained by smelting the ilmenite concentrates in the prepared form, done jointly by the VAMI, IMET AN SSSR and the Zaporozh'ye Ferroalloys Plant. At the same time, slags with a titanium dioxide content of over 80 per cent were obtained, with a fusion temperature of not more than 1600 degrees, and the electric power consumed amounted to 3000 kilowatt-hours per ton of slag. When the process is separated to be handled in two aggregates one can expect still more economical results.

When the VAMI, the IMET AN SSSR, and the Zaporozh'ye Ferroalloys Plant carried out this smelting, they took into account the experience of the TsNIIChM [Abbreviation not known, apparently the All Union Central Scientific Research Institute of Ferrous Metallurgy], the GIREDMET [Gosudarstvennyy nauchno-issledovatel'skiy institut redkiy metallov - State Rare Metals Scientific Research Institute], and other scientific research establishments.

At present, one of the titanomagnetite plants is producing slags by this technology with higher technical-economic indices.

Thus, the technology has been worked out for smelting titanium ores to slag with a high titanium content and the problems for research in the near future are: perfecting the existing technology by shortening the duration of melting, intensifying the reduction of iron oxides, decreasing the fusion temperature and (electric) conductivity of the slag, reducing heat losses, and securing a more complete utilization of the reducing capacity of the gases.

The projects being conducted by the IMET AN SSSR and the VAMI on two-stage smelting of ilmenite concentrate (zone reduction of iron oxides or in rotating body furnaces and melting the charge in an arc furnace, thus obtaining rich slags and metal with a low carbon content) and the work of the UFAN [Ural'skiy filial akademii nauk - Urals Branch of the Academy of Sciences, USSR] on the use of agglomerate mixed with flux obtained by sintering the charge mixture of ilmenite concentrate and limestone in electric smelting constitute the most promising trends in improving the technology of smelting titanium slag.

The results of research conducted by V. V. Mikhaylov and his co-workers (UFAN) provide evidence that agglomeration [sintering] results in complete destruction of the ilmenite on account of its oxidation,



with formation of hematite, perovskite, ANOSOVIT [Transliterated, possibly anosovite], and other minerals. The reducibility of the agglomerates obtained by the UFAN is higher than the reducibility of the ilmenite concentrate.

The question of the possibility of using blast furnaces for smelting titanium ores, taking into consideration the achievements of recent years (the use of low-shaft blast furnaces which operate on air blasts enriched with oxygen and with natural or coke oven gas forced into the furnace) merits further study.

In order to understand the nature of the processes which take place in the smelting of titanium ores, it is necessary to start with the most important peculiarities of the oxides and of certain compounds of titanium in forming solid solutions on the basis of lattices:  $Ti_3O_5$ ,  $Ti_2O_3$ ,  $TiO$ ,  $TiO_2 \cdot 2H_2O$ ,  $TiO_2 \cdot H_2O$ , and others.

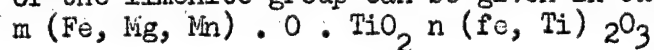
The basic physicochemical properties of these minerals -- melting point, toughness, electrical conductivity, the constants of equilibrium involving titanium compounds -- depend first of all on the composition of the solid solutions.

The process of forming solid solutions begins with the sintering stage or the smelting of ilmenite or titanomagnetite concentrates.

Thus, the sintering of fine crystalline titanomagnetites which have the structure of fine intrusions of ilmenite into the magnetite causes their complete recrystallization with disappearance of the natural silicate minerals and the formation of new phases: the solid solution of menatite in ilmenite, ANOSOVIT, fayalite, metallic iron, pseudobrookite, glasses, and others [18]. The process of forming a solid solution of hematite and ilmenite  $[Fe_2O_3]$   $FeO \cdot TiO_2$  takes place under weakly reducing or under neutral conditions with interaction of the silicate being melted with solid magnetite.

Iron metatitanate forms a continuous series of solid solutions with magnesium metatitanate (geikielite) and manganese (pyrophanite). Hematite is soluble in it to a limited extent. Thus, ilmenite has a complex composition. The natural color of the mineral is black, this is caused by a trace of trivalent titanium in it.

Studies carried out in the Institute of Metallurgy of the Academy of Sciences, USSR have established that metatitanates can form solid solutions in titanium sesquioxide [19]. Thus, titanium sesquioxide is included in the ilmenite mineralogical group. The general formula of minerals of the ilmenite group can be given in the following form:



Reduction of the iron oxides from a mineral of such complex composition depends upon the oxides of magnesium, manganese, and trivalent titanium contained in it. The reducibility of ilmenite is affected by its natural origin; thus, leucoxene ilmenite has a strongly-developed macroporous nature, a large reaction surface, and possesses the best reducibility.

The reduction of iron metatitanate ( $\text{FeOTiO}_2$ ) is a complicated process since, depending upon temperature conditions, reduction of iron metatitanate by solid carbon can give rise to the lower oxides:  $\text{Ti}_3\text{O}_5$ ,  $\text{Ti}_2\text{O}_3$ ,  $\text{TiO}$  and solidsolutions on the base of the lattices of these oxides.

A thermodynamic analysis of the reactions of the reduction of  $\text{FeO.tiO}_2$  by solid carbon (Table 2) permits one to conclude that the reduction of  $\text{FeO.TiO}_2$  to  $\text{Fe}$  and  $\text{TiO}_2$  or  $\text{TiO}$  is least probable. At the temperatures which prevail in the technological process, reactions which result in the formation of  $\text{Fe}$ ,  $\text{Ti}_3\text{O}_5$  (up to 1240 degrees) or  $\text{Ti}_2\text{O}_3$  (at higher temperatures) are most probable. The appearance of the lower oxides of iron in  $\text{Ti}_3\text{O}_5$  and of iron metatitanate in titanium sesquioxide ( $\text{Ti}_2\text{O}_3$ ), which makes the process of reducing the iron oxides more difficult.

TABLE 2

Reaction	Temperature (in degrees Kelvin)						
	1000	1100	1200	1400	1600	1800	2000
$\text{FeO.TiO}_2 + \text{C} = \text{Fe} + \text{TiO}_2 + \text{CO}(1)$							
$\Delta F = 37910 - 33.88 T$ calories	4030	640	92750	-9520	-16300	-23070	-2985
$3/4 \text{FeO.TiO}_2 + \text{C} = 3/4 \text{Fe} + \text{Ti}_3\text{O}_5 + \text{CO}(2)$							
$\Delta F = 40.06 - 36.39 T$ calories	3720	80	-3560	-10840	-18120	-25400	-32670
$2/3 \text{FeO.Ti}_2\text{O}_3 + \text{CO} = 2/3 \text{Fe} + \text{Ti}_2\text{O}_3 + \text{CO}(3)$							
$\Delta F = 42434 - 36.87 T$ calories	5560	1880	-1810	-9180	-16560	-23930	-3131
$1/2 \text{FeO.TiO} + \text{C} = 1/2 \text{Fe} + \text{TiO} + \text{CO}(4)$							
$\Delta F = 53684 - 37.62 T$ calories	16660	12300	8540	1020	-6510	-14030	-21560

Research conducted by A. V. Rudneva, with the participation of the authors of this paper, showed that when ilmenite concentrate was smelted in the form of ore-carbon briquettes, titanium sesquioxide appeared with over 80 per cent titanium dioxide content in the slag. When agglomerates made of Kusa ilmenite concentrates were smelted, the formation of solid solutions in titanium sesquioxide was observed with a content of the lower iron oxide. This is explained by the earlier appearance of the liquid phase, since the agglomerates have a melting point which is 100 - 150 degrees lower than that of  $\text{FeO} \cdot \text{TiO}_2$  (1380 degrees). Thus, the formation of the liquid phase makes possible a large diffusion of ferrous oxide in titanium sesquioxide.

The formation of solid solutions and the appearance of the liquid phase create conditions for the existence of ferrous oxide, and the dioxide and lower oxides of titanium, which lowers the activity of the ferrous oxide and makes the process of its reduction more difficult. The formation of the liquid phase, due to the development of processes of initial slag formation, introduces an additional complicating factor -- a molten product is formed which contains new constituents that again lower the activity of ferrous oxide.

Mineralogical studies of samples of the molten product taken from the furnace at different times up to the tapping of cast iron and slag which were conducted by the Institute of Metallurgy showed that during melting the processes of reducing titanium dioxide with the formation of  $\text{Ti}_2\text{O}_3$  and  $\text{Ti}_2\text{O}$  proceed more rapidly than the reduction of ferrous oxide to the metallic phase.

This causes the appearance of iron orthotitanate ( $2\text{FeO} \cdot \text{TiO}_2$ ) in the samples of the slag, along with the typical titanium minerals, but this disappears as an independent phase toward the end of melting.

When smelting agglomerates made of titanomagnetite concentrates with a continuous process with a closed top [of the furnace], the processes of reducing the iron oxides are completed basically in the region where the charge is in a plastic state. In the region near the electrodes molten slag enriched with iron oxides is formed since the melting of the charge proceeds at a faster rate than the process of reducing the iron oxides.

The composition of the final slag is a result of concentration of the previously formed slag on the hearth of the furnace where the processes of reducing the iron oxides and the continuously incoming ferrous slag formed near the electrodes have progressed further.

Consequently, the ferrous oxide content in the slag is determined by the ratio of the development of the processes of reducing the iron oxides in the slag and the melting of the charge in the vicinity of the electrodes. This relationship can be controlled by the value of the voltage and the periodicity of slag tapping. The best technological indices [performance] are achieved by working with comparatively low voltages with little fluctuation, which is especially important when one is using continuous melting with a closed top. Voltage values and periodicity of slag tapping are tested experimentally. Good preparation of the charge, particularly in the form of the charge, particularly in the form of ore-carbon briquettes, permits raising the voltage during smelting.

Thus, the development of the processes of reducing the iron oxides in the molten slag determine the productivity of the furnace. To intensify the smelting it will be necessary to develop to a maximum the processes of reducing the iron oxides prior to the transition of the charge to the molten state, which can be achieved by improving the contact of the reducing agent with the iron oxides (joint bruqietting), some deepening of the charge for more complete use of the reducing capacity and the heat content of the outgoing gases. Let us see which of these methods is most effective.

a. The Reduction of Ilmenite with a Gaseous Reducing Agent and Solid Carbon

Research on the reducibility of ilmenite by the gaseous reducing agent (carried out in the Institute of Metallurgy by F. B. Khalimov and the authors) carbon monoxide and hydrogen, conducted at different temperatures, showed that a constant increase is observed up to 1100 degrees in the reducibility of ilmenite and the constant of reducibility obeys Arrhenius' kinetic law. A deviation from this law is observed at temperature above 1100 degrees due to the development of slag formation.

Research [16, 20] has established that when ilmenite is heated, even in a stream of nitrogen, at 800 degrees one observes the decomposition of the solid solution with the formation of two of its modifications which differ from each other in external and other respects.

X-ray studies have established that the dark product of decomposition is a very fine mixture of ilmenite and magnetite [20]. In this case the reduction of iron begins in sections of the formation of magnetite.

The process of reducing agglomerates obtained from titanomagnetite concentrates has a more complex character (Figure 1).

The straight lines [linearity] of the reducibility of the agglomerates (Figure 1) show a change at temperatures about 800 degrees and at the extent of reduction of the iron at 27 - 35 per cent; in this case the higher the ilmenite content of the agglomerate (solid solution of hematite in ilmenite), the more pronounced this change is. When the temperature is increased, the change is shifted toward lowering the extent of reduction.

The physical properties of the agglomerates (porosity and size of the reduced grain) have a noticeable effect on the shifting of the change in the straight line of reducibility. Increasing the density has the same effect as increasing the temperature -- it shifts the change toward lower percentages of reducibility.

Study of polished sections under the microscope made is possible to establish that up to 800 degrees only the magnetite is reduced and that the solid solution of hematite and ilmenite is not reduced.

At higher temperatures, only the magnetite is reduced prior to the change while the magnetite and the solid solution of hematite and ilmenite are jointly reduced following the change. It is known that carbon dioxide molecules are large, thus when they are absorbed, they form droplets in the pores of the agglomerate. As the grain size, this is increased, they way is improved for diffusion of the reducing agent and the gaseous product of reduction. Since the back diffusion of the carbon dioxide proceeds at a somewhat slower rate than the forward diffusion of the carbon monoxide, this facilitates an increase in the partial pressure of the carbon dioxide in the reaction zone. As this reaction moves into a grain of the material, the partial pressure of the carbon dioxide decreases on its peripheral parts. As a result of this, the solid solution of hematite in the ilmenite begins to be reduced, which brings about an increase in the angular coefficient of the straight line of reducibility (the constant of reducibility) after the change.

When titanomagnetites are reduced by a solid carbon, the carbon dioxide formed in the zone of reaction is regenerated into carbon monoxide, which creates more favorable conditions for the intensive reduction of iron oxides.

A study of the reducibility of ilmenite and titanomagnetites by solid carbon permits one to conclude that the process of reducing iron oxides proceeds intensively at temperatures over 1200 degrees. In an intimate mixture of carbon and concentrate (in briquettes), the iron oxides are reduced by 60 per cent in ilmenite and by 80 per cent in titanomagnetite in 5 minutes.

The processes of primary slag formation do not have an inhibiting effect on the reduction of iron oxides for temperatures over 1300 degrees, that is, very close to the melting point for iron metatitanate (1380 degrees), (Figure 2).

b. The extent of Reduction by a Gaseous Reducing Agent (Indirect Reduction) in the Smelting of Titanium Ores.

The proper charging of carbon during smelting of titanium ores is an important condition for their successful smelting. This is possible on condition that the extent of the development of the reduction of iron oxides by waste gases is properly evaluated.

Research conducted by the authors with the participation of the engineer V. I. Solov'yev established that when agglomerates are being smelted from titanomagnetite concentrates, about 15 per cent of the total oxygen is removed by indirect reduction and about 21 per cent in the smelting of briquetted ilmenite concentrates. When the capacity of the furnace is increased, the extent of development of indirect reduction is increased, which in turn requires corrections in the amount of reducing agent in the charge. The expenditure of electric power is reduced by 5-10 per cent due to the development of processes of indirect reduction.

The direct reduction of iron oxides in electric smelting of titanium ores has seen a greater development (1.6 - 2.2 times) than blast furnace smelting of ordinary charges that do not contain ilmenite.

In shaft furnaces, the region of moderate temperatures, 600 - 1100 degrees, is not developed to a significant extent. As the height of the furnace is increased, one observes a sharp transition from moderate temperatures to temperatures of 1300 degrees and higher, at which the reduction of iron oxides proceeds intensively. Consequently, in electric smelting, the ratio between the development of the processes of direct and indirect reduction shifts in favor of direct reduction. It is seen from what has been said that intensification of the processes of reducing iron oxides can be achieved by intelligent increase in the height of the shaft of the furnace and mainly by ensuring intimate contact between the reducing agent and the concentrates in the form of ore-carbon briquettes. In this case the processes of primary slag formation do not have an inhibiting effect on the processes of reduction.

A transition to slags of optimal composition with moderate melting temperatures and low electrical conductivity consumption constitutes the second condition for increasing the productivity of an aggregate and reducing the consumption of electric power.

#### c. Studies of the Properties and the Phase Composition of Titanium Slags.

Both in the blast furnace as well as in the electric furnace, the final composition of the slag differs from that of the slags formed during the course of melting in the upper layers of the blast furnace (the bosh) and in the vicinity of the electrodes in electric furnaces. Moreover, the processes of initial (primary) slag formation play a decisive role in ensuring an even course of reducing smelting. In the smelting of titanomagnetites in blast furnaces, they determine the descent of the charge and the resistance to the flow of gases in the lower part of the furnace, and in smelting in ore-reducing electric furnaces -- the ferrous oxide content of the slag that is tapped.

Knowledge of the FeO -- TiO<sub>2</sub> diagram acquires great importance in understanding the character of primary slag formation in the smelting of titanium charges.

Research done by the Academy of Sciences, USSR on this system revealed three compounds with their melting points: 1375 degrees, 2FeOTiO<sub>2</sub> (iron orthotitanate); 1385 degrees, FeOTiO<sub>2</sub> (KRICHTONIT /Possibly crichtonite/; and more than 1450 degrees, FeO<sub>2</sub>TiO<sub>2</sub> (ferrous ANOSOVIT /Possibly anosovite/, (Figure 3).

Under oxidizing conditions ferrous oxide is not stable and is transformed into magnetite, therefore the compound FeO<sub>2</sub>TiO<sub>2</sub> is not formed. Under reducing conditions the appearance of the liquid phase in this system is observed at a temperature of 1100 degrees [21], if the FeO<sub>2</sub>TiO<sub>2</sub> concentration reaches 70 per cent.



Ferrous titanium slags have a low melting point. The character of crystallization of the slags is "short"; solidified titanium slag is melted at a higher temperature than its freezing point in view of the fact that titanium slags are not in equilibrium and their composition changes with time. Temperature fluctuations in the upper part of the blast furnace are undesirable in the smelting of titanium slags as they lead to thickening of the primary slags or to their crystallization, which interferes with the descent of the charge and disrupts the work of the furnace. The stability of the primary slags can be increased by introducing alkaline additives (soda and sodium chloride) [15, 16] which catalyze the process of reducing iron oxides [16, 20].

As a result of the interaction of soda or other alkaline compounds with ilmenite, free ferrous oxide is formed which is more easily reduced than iron metatitanate and iron titanate ( $\text{Na}_2\text{O} \cdot 0.3\text{TiO}_2$ ) [15, 16].

The addition of oxides of calcium, magnesium, or barium also exert a noticeable effect on the reduction of ilmenite, due to its destruction and the formation of metatitanates of calcium (perovskite), magnesium, and barium. However, this process is developed at temperatures over 1300 degrees [17].

Studies made by the authors and the engineer V. I. Solov'yev established that addition of oxides of the alkaline - earth metals decreases the final ferrous oxide content of the slag, but when the additives are increased, particularly calcium oxide, the rate of reduction of ferrous oxide in the molten slag is decreased.

The final titanium slags, as compared with the primary slags, always have a lower content of iron oxides and a higher content of titanium dioxide and a high melting point. Titanium slags are, I dare say, the most fluid slags.

Titanium is found in titanium slags in tetravalent, trivalent, and divalent form. The tetravalent titanium ion, like the silicon ion, is inclined to forming the complexes of  $\text{TiO}_6^{-8}$ , which form the structure of ANOSOVIT in the solid state [22].

Trivalent and divalent titanium ions obviously do not form complexes in the molten product and usually exist in the form of the simplest cations. Consequently, molten titanium slags do not contain complexes of the  $\text{Si}_x\text{O}_y$  type which are characteristic of molten silicates; therefore, when in the liquid state titanium slags have little viscosity [17].

Titanium slags have a large capacity for crystallization (a high rate of crystallization in a very narrow range of temperatures), and even as they are being tapped from the furnace they contain solid particles of such high-temperature minerals as ANOSOVIT, titanium, sesquioxide and other minerals. When being tapped, titanium slags remind one of ice breaking up on a river.

The high crystallization capacity of titanium slags is determined by the simplicity of the ion complexes in the liquid state which form, without special rearrangement, solid solutions on the base of the  $\text{Ti}_3\text{O}_5$  and the  $\text{Ti}_2\text{O}_3$  lattices in the solid state.

d. The Effect of the Lower Oxides of Titanium on the Fusability of Titanium Slags

Titanium slags should be separated into three groups in accordance with the influence of the lower oxides of titanium on their fusability, namely: slags which contain up to 50, 75, and above 80 per cent of titanium dioxide.

Slags of the first type are formed when melting titanomagnetites and their fusability, like that of blast furnace silicate slags, is determined essentially by the  $\text{CaO}$ ,  $\text{SiO}_2$  ratio up to the point where the  $\text{Ti}_2\text{O}_3$  :  $\text{TiO}_2$  is not above 0.5 [17]. With a greater value for this ratio, the fusability of the slag increases sharply on account of the formation of the lower oxides of titanium in it which are fused at quite high temperatures. According to Kubashevskiy's data [23],  $\text{Ti}_2\text{O}_3$  fuses at 2130 degrees and  $\text{TiO}$  at 2020 degrees.

The melting point of ANOSOVIT depends on the isomorphic admixtures dissolved in it and the relationships of titanium ions of different valences contained in the mineral.

K. Kh. Tagirov showed in his works that the melting point of ANOSOVIT can be changed noticeably by the ratios of the titanium oxides in it (Table 3).

TABLE 3

$\text{Ti}_2\text{O}_3$ : $\text{TiO}_2$	Melting Point (Degrees Centigrade)	Remarks
-----	1820	Titanium dioxide.
0.26	1720	ANOSOVIT, separated from slag and containing about 30 per cent $\text{TiO}_2$ .
0.30	1700	ANOSOVIT, separated from slag and containing about 40 per cent $\text{TiO}_2$ .
0.48	1670	Same as above

Infusability and thickening of titanium slags is often explained by development of processes of formation of titanium carbide [24, 25, 26]. However, it must be noted that thickening of titanium slags occurs before titanium carbide appears in them. Work done under the guidance of K. Kh. Tagirov showed that thickening of titanium slags is not explained by the formation of titanium carbide, which was not actually discovered, but by the presence of lower oxides of titanium, and first of all, by the high melting point of ANOSOVIT [27].

Engineer I. A. Karyazin studies the effect of lower oxides of titanium in the presence of ferrous oxide on the fusability of slags in the Institute of Metallurgy of the Academy of Sciences, USSR (Figures 4 and 5).

This research showed that a titanium sesquioxide content up to 10 per cent increased the fusability of slags in regions with low ferrous oxide content, but did not have any noticeable effect on the fusability of the slags where there was a 10 - 30 per cent titanium sesquioxide content. Addition of magnesium oxide was useful up to the formation of titanite in lean titanium slags and magnesium ANOSOVIT in slags rich in titanium since in the latter case the magnesium oxide, replacing the  $TiO$  in the ANOSOVIT, combined with the titanium dioxide. Further increase in the amount of magnesium oxide introduced into the slag caused the formation of BAYKOVIT [Transliterated, possibly baikovite] ( $MgO \cdot Ti_2O_3$ ) and magnesium orthotitanate ( $2MgO \cdot TiO_2$ ) which sharply increase the melting point of titanium slags.

Figure 5 shows a system analogous to the system shown in Figure 4, but with a higher constant magnesium oxide content. A comparison of these diagrams permits one to conclude that addition of magnesium oxide in regions containing little titanium sesquioxide decreased the fusability of titanium slags while its addition to regions with high  $Ti_2O_3$  content increased their fusability.

Titanium slags containing over 80 per cent  $TiO_2$  were formed in the smelting of rich titanium concentrates with adding flux to the charge and with up to 2 per cent ferrous oxide left in the slag.

Studies of the effect of the lower oxides of titanium on the properties of plant slags showed that a 20 - 25 per cent  $Ti_2O_3$  content in the slag did not cause any material complications in the smelting process and its melting point was not changed significantly.

e. The Effect of the Oxides of Potassium and Magnesium on the Fusability of Titanium Slags.

Under reducing conditions, the  $CaO - TiO_2 - Al_2O_3$  system has a wide field of readily fusable titanium slags with a melting point of 1450 - 1500 degrees with over 80 per cent titanium dioxide content, which is explained by the formation of an eutectic ANOSOVIT and perovskite. ANOSOVIT with an isomorphic admixture of alumina was discovered for the first time in this system.

Study of this diagram permitted a somewhat different treatment of the effect of alumina on the fusability of titanium slags. Sigurdson's [SIGURDZON] views, which are widespread in the literature [21], were developed when studying titanium slag systems under oxidizing conditions. Sigurdson discovered a wide field of readily fusable titanium slags and established that alumina increased fusability and shortened the region of existence of liquid slags, particularly in the limits of 5 - 20 per cent concentration of alumina. These conclusions are valid only under oxidizing conditions, when the compound  $Al_2TiO_5$  is

formed in the system with a melting point of 1850 degrees [28]. Under reducing conditions, however, the addition of 5- 20 per cent of alumina really does not increase the fusability of titanium slags since the introduction of alumina into the lattice of ANOSOVIT lowers its melting point.

In the  $\text{TiO}_2$  --  $\text{Al}_2\text{O}_3$  --  $\text{CaO}$  and  $\text{MgO}$  system, with a constant 10 per cent magnesium oxide content, it was established that the magnesium oxide decreased the field of readily fusable slags.

There is a relationship between the ratio  $\text{MgO} : \text{CaO}$  and  $\text{Ti}_2\text{O}_3$  in  $\text{TiO}_2$  in slags -- increasing the amount of magnesium oxide ( $\text{MgO} : \text{CaO}$ ) brings about a reduction of titanium dioxide to the sesquioxide ( $\text{Ti}_2\text{O}_3 : \text{TiO}_2$ ).

#### f. Electrical Conductivity of Titanium Slags

The electrical conductivity of titanium slags is an important technological property inasmuch as its value determines the immersion of the electrodes in the molten charge, and, consequently, the conditions of its natural heating through convection, which creates the conditions for smooth working of the furnace [29].

Studies made in the Institute of Metallurgy of the Academy of Sciences, USSR established that the electrical conductivity of titanium slags in which the titanium oxides are crystallized in the form of ANOSOVIT have a mixed, in a number of cases an electronic character.

In titanium slags which do not contain ANOSOVIT and which have the very same ratio of the remaining components, electrical conductivity increases with increases in temperature and apparently has an ionic character.

In the  $\text{TiO}_2$  --  $\text{SiO}_2$  --  $\text{Al}_2\text{O}_3$  --  $\text{CaO}$  --  $\text{MgO}$  system, increasing the titanium dioxide content in the charge from 0 up to 60 per cent increases the electrical conductivity from 0.1 to about mho per centimeter.

Increasing the amount of the lower oxides of titanium increases its conductivity and decreases the dependence of temperature on electric conductivity.

It is known from practice in smelting titanium ores in electric furnaces that addition of calcium oxide to the charge leads to a decrease in the melting point and the electrical conductivity of the slag, which permits better heating of the charge and creating conditions for smooth working of the electric furnaces.

In this connection, it is interesting to compare the effect of calcium oxide and of magnesium oxide on the electrical conductivity of the slags. Magnesium oxide contents up to 10 per cent have almost no effect on the electrical conductivity of slags, but sharply increase their conductivity at higher amounts. Thus, calcium oxide is the most suitable additive since it lowers the melting point and the conductivity of the slag charge.

g. The Principal Titanium Minerals in Titanium Slags and Their Technological Significance.

When studies were made on titanium slags in the Institute of Metallurgy of the Academy of Sciences, USSR, new groups of titanium minerals were discovered: solid solutions on the base of the lattice of  $Ti_2O_3$  -- ANOSOVIT  $n \text{ } \left[ (Ti, Al, Fe) 2O_3 \cdot TiO_2 \right] .m \text{ } \left[ (Ti, Fe, Mn, Ni, Co, Mg) O 2TiO_2 \right]$ ; solid solutions based on the lattice of titanium sesquioxide  $(Fe, Mg, Mn) O.TiO_2 n (Fe, Ti) 2O_3$  solid solutions based on the lattice of orthotitanate --  $2(Mg, Fe) O.TiO_2$ ; BAYKOVIT --  $MgO.Ti_2O_3$  and IMANIT  $\left[ \text{Transliterated, possibly imanite} \right]$  --  $3CaO.Ti_2O_3.3SiO_2$  and others.

The composition of ANOSOVIT, as indicated in the formula, includes ions of all valences of titanium; divalent and trivalent titanium ions can be replaced, respectively, by ions of iron, magnesium, manganese, aluminum, and others. As stated in the literature, ANOSOVIT has been obtained with an isomorphic admixture of cobalt  $\left[ 30 \right]$ . ANOSOVIT has different optical properties, depending upon its composition.

The technological importance of ANOSOVIT consists of the fact that it determines the melting point (fusability) and its reactivity in sulfuric acid and chlorine. Therefore, it is possible to obtain slags with the best technological properties, both for the conditions applicable for smelting ilmenite in electric furnaces as well as for recoverability in acids, chlorine, and other possible processes, by changing the composition of the ANOSOVIT.

Minerals which are components of the solid solutions based on titanium sesquioxide take second place after ANOSOVIT in respect to prevalence in titanium slags, and occurs more frequently in slags with a high titanium dioxide content.

Solid solutions based on the lattice of  $Ti_2O_3$ , like ANOSOVIT, do not have a definite melting point. Titanium sesquioxide can exist in slags along with a high ferrous oxide content and obviously forms compounds of the spinel  $FeO.Ti_2O_3$  type which have not been discovered in titanium slags as yet.

The technological and optical properties of titanium sesquioxide, which crystallize in slags change within very wide bounds and are determined by its mineralogical composition.

Research done by the Institute of Metallurgy of the Academy of Sciences, USSR and the VAMI has established that the lower oxides of titanium, in the first instance the solid solutions based on  $Ti_2O_3$ , interact actively with the oxygen in the air, which causes the phenomenon of the friability of titanium slags.

In the process of oxidation, the slag forms a flaky structure, the color of the slag changes from black to a light brown, and the general titanium content in the slag, in terms of  $TiO_2$  is decreased:

		TiO <sub>2</sub> *	FeO
Melt A	Slag at output	90.4	1.28
	Upper part of the slag ingot	85.0	1.09
	Friable part of the slag	81.5	0.96
Melt B	Slag at output	82.63	2.60
	Upper part of the slag ingot	78.8	2.02
	Friable part of the slag	76.53	4.12

\* Total amount of titanium in terms of titanium dioxide.

X-Ray structural and mineralogical research has established that rutile (TiO<sub>2</sub>) is formed in the process of oxidizing titanium sesquioxide.

The solid solutions based on the lattice of orthotitanate, discovered in the Institute of Metallurgy of the Academy of Sciences, USSR, are formed in primary slags; in the process of reduction they are made lean by iron, being transformed into magnesium orthotitanate. Crystallization of magnesium orthotitanate in the final titanium slags indicates the destruction of the magnesite lining unless magnesium is introduced into the charge as a flux in the smelting process.

Magnesium orthotitanate impairs the reactivity of slags in acids. When the magnesium orthotitanate contained in titanium slags is chlorinated, magnesium chloride is formed, 40 - 50 per cent of which is carried away with FeCl<sub>3</sub> and AlCl<sub>3</sub>. These chlorides complicate the work of condensation systems -- the worst bottleneck of the entire technological process of chlorination. Addition of calcium oxide obtaining magnesia titanium slags improves their physical properties and facilitates the work of condensation systems in the chlorination of these slags since the quantity of calcium chloride is increased in melting chlorides, which sharply curtails the escape of the chlorides.

BAYKOVIT (MgO.Ti<sub>2</sub>O<sub>3</sub>) is a high - temperature mineral which crystallizes in titanium slags which contain significant quantities of magnesium oxide (more than 6 per cent) with a high Ti<sub>2</sub>O<sub>3</sub> : TiO<sub>2</sub> ratio. The composition of BAYKOVIT was determined provisionally for the first time by K. Kh. Tagirov and A. B. Rudneva and requires further clarification. It received its name in honor of the well known soviet metallurgical scientist Academician A. A. Baykov.

The technological significance of BAYKOVIT requires further clarification. This mineral was discovered in slags with a high magnesium oxide content in smelting work was being done in the Zaporozh'ye Ferroalloys Plant jointly with the VAMI and the Institute of Metallurgy of the Academy of Sciences, USSR.

IMANIT was discovered in the SiO<sub>2</sub> -- TiO<sub>2</sub> -- CaO system as an intermediate phase in the process of reducing titanium dioxide. The composition of IMANIT, which was named in honor of the Institute of Metallurgy of the Academy of Sciences, USSR was established for the first time by A. V. Rudneva.



Among the other titanium minerals the most important are -- perovskite ( $\text{CaO} \cdot \text{TiO}_2$ ), titanaugite  $m(\text{CaO} \cdot \text{MgO} \cdot 2\text{SiO}_2) \cdot n(\text{CaO} (\text{Al}, \text{Ti})_2\text{O}_3\text{SiO}_2)$ , titanite  $\text{CaO} \cdot \text{TiO}_2\text{SiO}_2$  and others. Perovskite is a stable chemical compound of calcium oxide and titanium dioxide. The formation of perovskite in the slag lowers its electrical conductivity and enables the furnace to work smoothly. The optimal composition of titanium slag is determined by proper selection of the ratio of the following mineralogical phases: ANOSOVIT, solid solutions based on titanium sesquioxide, and perovskite.

Titanaugite is widely prevalent in comparatively lean titanium slag is determined by proper selection of the ratio of the following mineralogical phases: ANOSOVIT, solid solutions based on titanium sesquioxide, and perovskite.

Titanaugite is widely prevalent in comparatively lean titanium slags with a 3 to 8 per cent magnesium oxide content. Titanaugite plays the role of a cement in the slag mass, and it improves the reactivity of titanium slags in acids. Titanium slags which contain titanagite have a higher viscosity and a lower melting point.

Titanite is formed in lean silicate titanium slags with a low magnesium oxide content (2 - 3 per cent) in it. It has no practical significance in slags with a high titanium dioxide content.

In concluding this examination of the properties of titanium slags and their phasal composition, one should note that if intensification of the processes of reducing iron oxides in the smelting of titanium ores should be regarded as the first condition for improving productivity, then the transition to readily fusable slags with low electrical conductivity, whose optimal composition is determined by the selection of the ratio of the ANOSOVIT, the solid solutions based on titanium sesquioxide, and perovskite which are crystallizing in the slag constitutes the second condition.

## II. THE CHLORINATION OF TITANIUM COMPOUNDS, TITANIUM SLAGS, AND CONCENTRATES

The classical works of D. I. Mendeleev set forth a method for obtaining  $\text{TiCl}_4$  by chlorinating in the presence of carbon as the reducing agent for the dioxide and other titanium compounds with gaseous chlorine.

Studies were started in 1916 with the aim of developing the technology of producing titanium tetrachloride in Russia. The experiments were conducted in two directions: chlorination of titanium carbide and chlorination of titanium-containing ores in the presence of solid carbon. The following worked on the first assignment: Professor Ye. V. Biron and S. I. Oreshkin in the laboratory of the Forestry Institute; M. S. Maksimenko, V. D. Nikol'skiy, and others in the Elektrosplav Plant; K. F. Beloglazov and Yu. F. Kriger in the laboratories of the Mining and Electrical Engineering Institutes.

Work on chlorination was preceded by the studies of M. S. Maksimenko on obtaining titanium slags and titanium carbide. M. S. Maksimenko was one of the first in Russia to smelt ilmenite concentrates with addition and without addition of flux and obtain cast iron and high-titanium slags.

According to the data from these studies, titanium carbide was very readily chlorinated, beginning with a temperature of 200 degrees, with simultaneous ancillary chlorination of iron oxides with formation of  $\text{FeCl}_3$ . The iron chloride clogged the condensation system, and the researchers then came to the correct conclusion of the need for the maximum decrease in the iron oxide content of the slags and concentrates.

In 1917 Russian research workers developed the technology for chlorinating ilmenite through titanium carbide and did research work on the chlorination of titanium dioxide which have not lost their importance even up to the present.

Important research on the chlorination of titanium compounds was done by Soviet scientists, both in the form of studies of the theoretical principles and the mechanics of the process of chlorination, and also on the development of technological processes which played a decisive role in establishing modern industrial methods for producing  $\text{TiCl}_4$ . This research work was done in the GIPKh [Gosudarstvennyy institut prikladnoy khimii - State Institute of Applied Chemistry], GIREMET, VAMI, and the Academy of Sciences, USSR (IONKh [Institut obshchey i neorganicheskoy khimii - Institute of General and Inorganic Chemistry], and the IMET [Institut metallurgii - Institute of Metallurgy] imeni A. A. Baykov).

#### a. The Theoretical Principles of the Chlorination of Titanium Compounds

In order to understand the essential features of the processes which take place in the chlorination of titanium raw material, knowledge of the properties of titanium chlorides, the physico-chemical principles of the chlorination of titanium compounds, and the secondary processes which go on during chlorination is important.

Titanium forms a number of compounds with chlorine:  $\text{TiCl}_4$ ,  $\text{TiCl}_3$ ,  $\text{TiCl}_2$ , whose basic properties are shown in Table 4.

TABLE 4

	$\text{TiCl}_4$	$\text{TiCl}_3$	$\text{TiCl}_2$
Specific gravity, grams per cubic centimeter	1,727	2.68	3.12
Viscosity at 20 degrees (in poises)	0.00836		

$V_t$  - specific volume

$$n_t = 0.000516$$

$$V_t = 0.5171$$

$$V_t = 0.56773 (1 + 9.6457 \times 10^{-4} t + 6.02 \times 10^{-7} t^2 + 5.94 \times 10^{-9} t^3)$$

Translator's note: Although this line is cited in the table of corrections in the back of the book, no  $t$  was inserted after the  $9.6457 \times 10^{-4}$  to read  $9.6457 \times 10^{-4}t$ . The term as it appears in the book makes no sense. I added the final parenthesis.

Vapor pressure

$$\lg P = -2919T^{-1} \quad \lg P = -10000$$

$$+ 5.788 \lg T + 25.129 \quad - 4.52 \lg T + 25.08$$

Free energy

$$\Delta F = -180700 - 1.8 T \lg T + 34.65 T \quad \Delta F = -128000 - 9.7 T \quad \Delta F = -12290 - 12.2 T \lg T - 68. T$$

Fusion point

- 25

730 degrees Sublimes

Boiling point

137 degrees

960 degrees 1000 degrees (under pressure)

General state at 20 Degrees

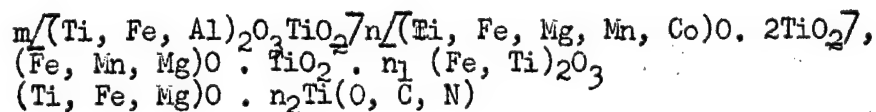
Colorless Liquid

Solid of violet color

Solid, Brown and brownish-black crystals.

Titanium forms three most prevalent oxides:  $TiO_2$ ,  $TiO_3$ , and  $TiO$  and a number of oxides of complex composition:  $Ti_3O_5$  [31],  $Ti_2O_3 \cdot 3 - 4 TiO_2$  [32], and others.

The following complex solid solutions are formed on the basis of titanium oxide lattices:

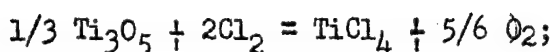


Study of the kinetics and the thermodynamics of chlorination of these solid solutions is of great practical and theoretical interest and is a subject for research at present.

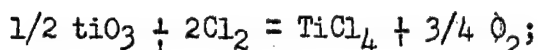
Chlorination of titanium oxides by chlorine gas proceeds in accordance with the reactions:



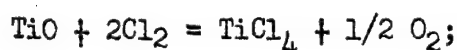
$$\Delta F = 37760 - 12,8 T;$$



$$\Delta F = 13425 - 5,12 T;$$



$$\Delta F = -1325 - 1,825 T;$$

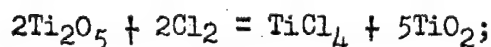


$$F = -180700 + 29,0 T.$$

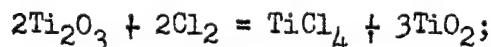
A comparison of the values of the free energy of these reactions at the temperatures of the technological process shows that the temperature at the beginning of chlorination of the titanium oxides decreases with a decrease in valence. In practice, only the oxide and the lower oxide of titanium are chlorinated directly by chlorine gas.

According to data in the literature, titanium dioxide begins to chlorinate at 800 degrees, however, this may be valid on condition that intermediate compounds between chlorides and oxides, oxychlorides of titanium, are formed in the products of chlorination. The extent of the development of such a reaction is insignificant and does not exceed 1 per cent in a period of 4 hours at a temperature of 800 degrees and a flow of the gas at 15 liters per hour [33].

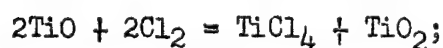
Chlorination of the lower oxides of titanium can proceed with the formation of titanium tetrachloride and titanium dioxide:



$$\Delta F = -108240 + 33,35 T;$$



$$\Delta F = -363170 + 73,7 T;$$



$$\Delta F = -154560 - 55,4 T;$$

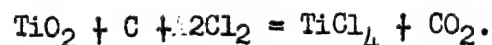
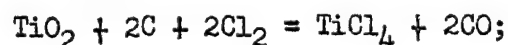
A comparison of the reactions in 1 - 4 and in 5 - 7 permit one to conclude that chlorination reactions with the formation of  $TiCl_4$  and titanium dioxide possess the most favorable conditions for development and thus their occurrence is more probable.

Studies of the kinetics of the chlorination of  $TiO$  made in the Institute of Nonferrous Metals and Gold [34] showed that the lower oxide of titanium was chlorinated only with the formation of titanium tetrachloride and titanium dioxide.

Chlorination of titanium oxides in the presence of a reducing agent (carbon) is of great practical importance since rutile and titanium slags are chlorinated in the presence of solid carbon.

The mechanism of the reactions of chlorination of titanium by solid carbon [Sic] was studied by the Soviet research workers A. V. Panfilov and co-workers [35, 36, 37, 40].

The process of chlorination of titanium dioxide can proceed according to one of these reactions:

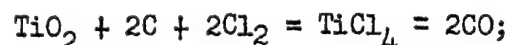


Reaction (8) predominates up to temperatures of 700 degrees, while only reaction (9) is of practical importance at temperatures around 1000 degrees.

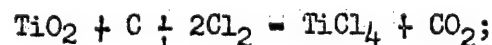
The intensity of chlorination depends not only on the type of compound, but also on the activity of the chlorine, since atomic chlorine interacts more effectively than molecular chlorine [33]. Phosgene, carbon tetrachloride, and others serve as activators of the chlorine. It was established experimentally that the interaction temperature of chlorine with titanium dioxide -- chlorine 800 degrees, phosgene 450 degrees, and carbon tetrachloride 430 degrees.

The literature contains recommendations in regard to technological processes for chlorination of titanium compounds by carbon tetrachloride and phosgene, but they have not become widespread [37].

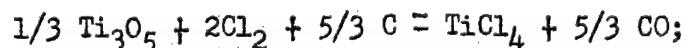
Let us examine the most important reactions of the chlorination of titanium oxides in the presence of carbon:



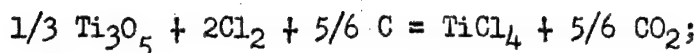
$$\Delta F^0 = 11560 - 54,60 T;$$



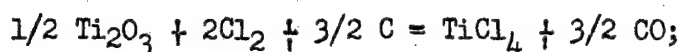
$$\Delta F^0 = - 56360 - 13,0 T;$$



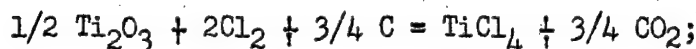
$$\Delta F^0 = - 31070 - 40 T;$$



$$\Delta F^\circ = 65075 - 5,3 T;$$



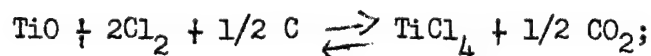
$$\Delta F^\circ = - 41375 - 33,18 T;$$



$$\Delta F^\circ = - 71970 - 2,0 T;$$

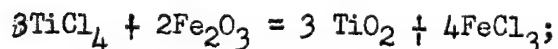
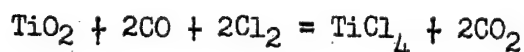


$$\Delta F^\circ = - 85100 - 13,3 T;$$



$$\Delta F^\circ = - 105,500 - 7,6 T.$$

In the opinion of some research workers, the role of addition of the reducing agent consists in the lowering the dissociation pressure of oxygen in the system, thus facilitating more complete chlorination. Thus, the role of the reducing agent amounts, so to speak, to reducing the oxide to a lower valance or to metal, which is more easily reduced. Undoubtedly, this process is really much more complicated. The character of the development of the process for chlorinating titanium oxides in the presence of solid carbon reminds one of the reduction of the oxides of solid carbon [38], and the mechanism of interaction can be presented as a summary process:



The composition of the gas phase in chlorination is determined by the rate of the reaction (19), which moves in the direction of forming carbon dioxide up to 700 degrees and in the direction of carbon monoxide at higher temperatures.

Knowledge of the degree of development of reaction (19) under the conditions of the chlorination of titanium oxides is of great practical importance in the proper measuring of the reducing agent in the charge. A large excess of the reducing agent leads to contamination of the chlorinator and premature stoppage of the units.

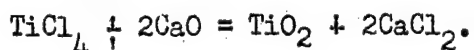
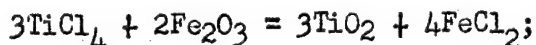


Research conducted by the Institute of Metallurgy of the Academy of Sciences, USSR established that the process of chlorination of titanium dioxide in the presence of carbon monoxide is developed to a significant extent only at temperatures above 900 degrees, and proceeds intensely at 1000 - 1050 degrees. Reactions of the chlorination of titanium dioxide in the presence of solid carbon take on noticeable development at lower temperatures than in the presence of carbon monoxide.

Studies on the problem of the transfer of chlorine and catalysis occupy an important place in the works of Soviet Scientists.

Under certain conditions, the chlorides themselves can be sources of chloring. Thus, the higher chlorides can dissociate and release chlorine gas which has a more active chlorinating action than molecular chlorine. Titanium tetrachloride, which can chlorinate iron oxides, alkali metals, and alkaline-earth metals, has such a property.

A. V. Panfilov and others [36] have pointed out the possibility of the occurrence of the following reactions.



According to these reactions, the secondary processes take place in the upper part of industrial chlorinators and the degree of their development, as in other processes in industrial units, have not as yet received the attention due them from research workers.

The addition of chlorides of the alkali and alkaline-earth metals to sulfates or phosphates of titanium can lead to exchange reactions with the formation of titanium tetrachloride [33].

Thus, the problem of the transfer of chlorine and exchange reactions leads directly to the problems of catalysis, chlorinating roasting, and secondary processes which may take place in chlorinators.

When titanium oxide is chlorinated with a small addition of an oxide of zirconium, cerium, or other rare - earth elements, the reaction is significantly developed at temperatures of 300 - 400 degrees [33].

The addition of 0.1 per cent of manganese peroxide with a 2 : 1  $\text{TiO}_2$  : C ratio leads to a noticeable acceleration of the chlorination reaction at 420 degrees [37].

Since chlorination is a heterogeneous process (the interaction of the chlorine and the oxides takes place on some surface or other), then increasing the reaction surface will facilitate the intensification of chlorination; preparation of the raw material with the purpose of increasing the porosity of the briquettes is of special importance.

#### b. The Chlorination of Titanium Slags

Titanium slags have a complex composition and chlorinating them presents a difficult problem. This difficult problem was first solved on a large industrial scale in the Soviet Union.

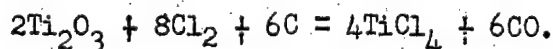
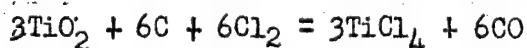
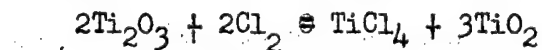
Beginning in 1930, projects were conducted on the chlorination of titanium compounds in the IONKh of the Academy of Sciences, USSR, the GIPKh, and the GIREDMET. The GIREDMET was first to carry out projects on the chlorination of titanium slags and this institute, jointly with the Podol'sk Chemico-Metallurgical Plant, mastered the industrial production of titanium tetrachloride by chlorinating its dioxide.

The research done by the VAMI and the Podol'sk Plant on the chlorination of titanium slags under industrial conditions resulted in the development and the introduction into industry of technology and facilities which would permit the production of high-quality titanium tetrachloride from slag.

At present, several methods have been recognized in our country and abroad for chlorinating titanium-containing materials which have been introduced into production or are under study. These methods include chlorination with chlorine gas or phosgene in shaft furnaces at temperatures of 300 to 1000 degrees, chlorination in a suspended state in a liquid or gaseous medium, and others. Two types of chlorinators have become prevalent in the production of titanium tetrachloride, of which one is a unit of shaft - type with batch operation and the other is a unit for zone chlorination -- for continuous operation [39].

Studies made on the chlorination of titanium slags in the Institute of Metallurgy of the Academy of Sciences, USSR showed that the lower oxides of titanium like  $Ti_3O_5$ ,  $Ti_2O_3$ , and  $TiO$  are chlorinated in slags according to reactions (I and II), forming titanium dioxide and titanium tetrachloride in the products. Reaction I is primary for chlorination of the lower oxides of titanium in the presence of solid carbon.

Thus, the chlorination of the lower oxides of titanium:  $Ti_3O_5$ ,  $Ti_2O_3$ , and  $TiO$ , using the chlorination of  $Ti_2O_3$  as an example, can be presented as the two-stage process:



The rate of progress of the summary process is determined by the velocity of the reaction of the chlorination of titanium dioxide in the presence of solid carbon; therefore, the chlorination of the lower oxides progresses at the same rate as the chlorination of titanium dioxide in the compared conditions. The lower temperature of the be-

ginning of the process of chlorinating the sesquioxide to form titanium dioxide and titanium tetrachloride is an advantage of the chlorination of the lower oxides.

It is pointed out in the works published by V. Kroll that flux additives like the oxides of calcium and magnesium decrease the rate of chlorination of the briquettes of the charge, covering them with calcium chloride and magnesium chloride. However, the significance of the fact is greatly exaggerated in these works as studies made by the VAMI and the experience of the titanium-magnesium plants show that the difficulties caused by flux additives amount to a small overexpenditure of chlorine and the need for periodic tapping of chlorides of the molten charge.

These difficulties can be overcome since the share of the chlorine in the cost of titanium is negligible [7&7] and tapping the chlorides of the liquified charge will not give rise to complications if the thermal conditions of the furnace have been properly set up for chlorination.

The use of slags with a small addition of flux (calcium oxide) in chlorination markedly decreases the power requirements of the process of producing titanium slags due to the transition to stable titanium slags of the ANOSOVIT - perovskite type with a melting point of 1400 - 1500 degrees. The addition of calcium oxide limits the reduction to the lower oxides and decreases the ferrous oxide content in the slags to 1 - 1.5 per cent, prevents friability in the titanium slags, and permits wide application of their wet granulation.

Chlorination of silica plays no small role in the chlorination of titanium slags since it accumulates, along with residual carbon, in the furnace, which leads to periodical stoppages. In the presence of the carbon free silica begins to chlorinate at 1200 degrees. In the combine state, silica is chlorinated along with those oxides whose silicates are subject to chlorination. In titanium slags which contain titanaugite as the principal silicate mineral, the silica is completely chlorinated. Therefore, the flux additives should be introduced with the purpose of limiting reduction of titanium dioxide to the lower oxides and forming a silicate phase in the slag -- titanaugite.

In the chlorination of titanium slags, the condensation stages are decisive for the entire technological process of chlorination. The principle difficulties are linked with the development of designs for production condensations systems. The VAMI and the GIREDMET, together with the Podol'sk Plant, have developed satisfactory designs for condensation systems, but studies are being continued in this field.

The principal difficulties encountered in the operation of condensation systems are caused by: a) clogging the condensation system by solid chlorides of iron, aluminum, magnesium, and complex compounds of titanium tetrachloride with other chlorides; b) very fine dust and solid reaction products which have been only partially removed by cyclones or dust chambers falling into the condensate; c) clogging of

the pipe lines with a mixture of dust, liquid tetrachloride, titanium oxychloride, and compounds of more complex composition; d) contamination of the gas pipe lines with products of the hydrolysis of chlorides; e) destruction of the condensation system due to the interaction of iron with hydrochloric acid vapors, and others.

Facilities of different types are used for condensation; the selection of designs is determined essentially by the composition of the raw material used in chlorination [397].

The titanium tetrachloride obtained is purified by removal of such admixtures as chlorine, phosgene, chlorides of silicon and vanadium, oxychlorides of titanium, vanadium, chrome, and others. The solubility of certain chlorides in titanium tetrachloride is low; for example, the solubility of ferric chloride is merely 0.2 per cent. The majority of the chlorides and oxychlorides are separated from  $\text{TiCl}_4$  by filtration.

In order to lower the solubility of chlorides (admixtures of titanium tetrachloride) they are subjected to reduction and are converted to salts with a high boiling point (the lower chlorides) or to complex ashes which are insoluble in  $\text{TiCl}_4$ . The following are utilized as reducing agents: powdered metals, for example, copper, metal hydrides, amalgams, chlorides of the alkaline-earth metals, iodides, organic compounds, soap, fats, tars, hydrogen, sulfide, and others. Copper and hydrogen sulfide are most often used as reducing agents.

Admixtures with low boiling points and soluble gases are removed by rectification of titanium chloride.

Studies made by the VAMI showed that noticeable quantities of titanium oxychloride ( $\text{TiOCl}_2$ ), whose solubility increases with the temperature are dissolved in titanium tetrachloride and that it can be removed by rectification or vacuum distillation.

Study of the following systems is of great importance in the problem of condensing and purifying titanium tetrachloride:  $\text{TiCl}_4$  --  $\text{FeCl}_3$ ;  $\text{TiCl}_4$  --  $\text{AlCl}_3$ ;  $\text{TiCl}_4$  --  $\text{VOCl}_3$  --  $\text{VCl}_3$  and others. I. S. Morozov is carrying on work in the IONKh of the Academy of Sciences, USSR to decrease the amount of organic compounds with a carboxyl group and soluble hydrogen chloride contained in the tetrachloride. The VAMI is conducting work on the molecular spectroscopy of titanium tetrachloride.

#### c. The Chlorination of Titanium Concentrates

A. V. Panfilov and Ye. G. Shtandel' [367] studies the chlorination of ilmenite and titanite concentrates. Ilmenite of the following composition was used in the experiments:

TiO	FeO	Fe <sub>2</sub> O <sub>3</sub>	SiO <sub>2</sub>	Cr <sub>2</sub> O <sub>3</sub>	MgO
43.92	31.41	16.85	1.84	0.05	7.76

and Titanite:

SiO <sub>2</sub>	TiO <sub>2</sub>	CaO	FeO	Fe <sub>2</sub> O <sub>3</sub>	MgO	MnO	P <sub>2</sub> O <sub>5</sub>	H <sub>2</sub> O
26.83	25.42	28.45	1.32	2.98	1.29	0.13	7.18	0.14

These experiments showed that chlorination of ilmenite in the presence of a reducing agent proceeded at 5.57 per cent at 300 degrees and 95.58 per cent at 740 degrees (in one hour with delivery of chlorine gas at the rate of 2 liters per hour). With direct chlorination of ilmenite, as in the case of slags with a high iron content (of the type used in the Sorel Plant), along with the difficulties encountered in organizing uninterrupted operation of the condensation system, there arises the problem of extracting chlorine which is combined in the form of iron chlorides -- a problem that has not been solved at present [7].

The chlorination of titanite [sphenite] concentrates takes place at higher temperatures than ilmenite, and chlorination of calcium oxide and silica (chlorination was done in the presence of carbon) takes place along with chlorination of titanium dioxide. Calcium oxide is chlorinated most readily, then titanium dioxide, and finally silica. At 800 degrees all the calcium oxide and the titanium dioxide is chlorinated and 60 - 70 per cent of the silica.

Titanite is a poor raw material as far as titanium is concerned. The enormous expenditure of chlorine and the limited possibilities for using calcium chloride are the principal obstacles in the use of titanite concentrates as raw material for producing titanium.

### III. PRODUCTION OF METALLIC TITANIUM

the industrial technological methods for producing sponge titanium are based on metallothermal methods for reducing titanium tetrachloride by magnesium and sodium. The production of pure titanium by refining metallic titanium with the iodide method under industrial conditions was mastered for research and special projects. Other methods for producing titanium have not as yet passed out of the stage of laboratory or pilot plant research. These include: a) metallothermal methods for reducing titanium dioxide with calcium or calcium hydride or by multistage reduction with magnesium and calcium; b) processes based on the decomposition of the lower chlorides of titanium obtained by reducing titanium tetrachloride with aluminum, sodium, and other reducing agents; c) electrochemical methods for producing titanium --

electrolysis of chloride, oxide and fluoride compounds, and the refining of titanium ligatures and scrap from titanium alloys, and others.

In the Soviet Union titanium was first produced in small quantities for the needs of the electrical engineering industry.

A comparative study was made in the GIREDMET of different methods for producing metallic titanium: reducing titanium dioxide and titanium tetrachloride with sodium, calcium, magnesium, and other reducing agents.

In order to produce titanium suitable for construction purposes, magnesium thermal method for reducing titanium tetrachloride was selected.

This method was tested experimentally, then the production of titanium on an industrial scale was mastered at the Podol'sk Chemico-Metallurgical Plant.

In the USSR at the present time, the VAMI has conducted and is conducting large research projects on perfecting the technology of producing titanium, on developing apparatus of large capacity and new types of apparatus, as a result of which the essential data have already been accumulated for creating a large titanium industry in the USSR based on the magnesium thermal method.

The second type of projects on producing metallic titanium in the USSR was begun in 1947 by the TsNIChM. After a comparative study of a number of methods was made, a method for reducing titanium dioxide with calcium hydride was adopted. This method produces a finely dispersed powder which is converted to a compact state by powder metallurgy and suitable processing. The titanium produced by this method has mechanical properties that are somewhat poorer than those of titanium produced by the magnesium thermal method.

#### a. The Reduction of the Oxides and the Halides of Titanium

Dissociation of titanium oxides into metal and oxygen are scarcely probable in practice -- the equilibrium constant of dissociation of titanium dioxide at 2500 degrees Kelvin is equal to  $4 \times 10^{-5}$  atmospheres, the monoxide  $1 \times 10^{-5}$ , and only at temperatures of 5000 Kelvin is the constant equal to 10. Gaseous reducing agents do not reduce oxides of titanium and only atomic hydrogen has any prospects in this respect; it yields the same results as metallic calcium in the reduction of titanium dioxide, but its effectiveness as a reducing agent is decreased sharply with an increase in temperature [41, 42]. When titanium dioxide is reduced with carbon, the reduction produces titanium carbide.

Of the metallic reducing agents only metallic calcium is capable of decreasing the oxygen in the titanium down to practically permissible limits. It was established experimentally that the oxygen dissolved in titanium is in equilibrium with the calcium oxide and liquid metallic calcium in concentrations, at 800, 900, and 1000 degrees, in percentages, respectively: 0.049, 0.066, and 0.136 [47]. Berillium



and magnesium are in equilibrium with 2.3 and 6.6 per cent of oxygen dissolved in titanium.

The literature contains references starting that thorium and gadolinium are the best deoxidizers, but that when gadolinium is used, the equilibrium concentration of oxygen is 0.1 per cent [42]. Of the metallic reducing agents, only metallic calcium can be used for reducing titanium dioxide, but it usually contains quite large quantities of nitrogen in solution, therefore metallic calcium must be distilled prior to utilization.

The existing methods for obtaining titanium which are applicable in practice specify two or more stages of reduction of the titanium oxides, respectively for calcium - magnesium and for other reducing agents.

The process of reducing titanium dioxide with metallic calcium takes place with the release of 158 kilocalories per gram of products and reagents. This heat is not sufficient to produce the metal in compact form. Intensifiers are added to the charge to increase the specific heat effect of the process. According to preliminary studies made by the UFAN sulfur is such an additive which strongly increases the exothermality of the process and metallic titanium is produced in a compact form.

When titanium dioxide is reduced with silicon, titanium silicide is formed, and when aluminum is used, titanium-aluminum alloys are formed. The oxygen content in a titanium-aluminum alloy is increased with an increase in the titanium in it; thus, with a 43 per cent Ti content, the oxygen in the alloy amounts to 0.1/0.13 per cent; with 64 per cent Ti -- up to 4 per cent; and with 85 per cent Ti -- about 11.5 per cent oxygen [42]. The titanium can be extracted from the alloys obtained by means of electrolytic refining. This method of refining titanium wastes was developed first by the TsNIChM.

Metallic titanium can be obtained by dissociation of the chlorides and bromides. However, this process requires high temperatures for its progress. Of all the halogens only titanium tetraiodide decomposes at lower temperatures.

This method, despite all its simplicity, is expensive, and the metal iodide is used in limited quantities.

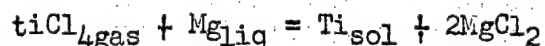
In the USSR the iodide method of refining titanium was developed in the GIREDMET and it was mastered industrially in the Podol'sk Chemico-Metallurgical Plant.

The Titanium most free of oxygen can be obtained in industrial quantities by reducing titanium tetrachloride. The reduction of  $TiCl_4$  with hydrogen is of no practical significance since the equilibrium constant of this reaction is at 1000 degrees Kelvin, 1500 degrees Kelvin, and 2000 degrees Kelvin --  $1 \times 10^{-13}$ ,  $1 \times 10^{-6}$ , and  $2 \times 10^{-3}$  respectively. There are indications that when titanium tetrachloride is reduced by atomic hydrogen pure metallic titanium can be obtained [42].

Titanium tetrachloride is a very stable chemical compound; thus, when it is heated to 5500 degrees Centigrade, one does not observe any noticeable dissociation of this compound. The lower chlorides are unstable when heated and readily convert to titanium tetrachloride and a very dispersed metallic powder. Titanium dichloride has such properties. As for  $TiCl_3$ , it usually dissociates to titanium tetrachloride and titanium dichloride. At present studies are being made on the decomposition of the lower chlorides; however, the most important problems which require solutions are: preventing the oxidation of pyrophoric titanium powder, protecting vacuum pumps from the corrosive action of  $TiCl_4$ , and obtaining lower chlorides of satisfactory purity. The decomposition of the lower chlorides into the metal and tetrachloride is very complicated process and the prospects of its utilization are very limited.

#### b. The Reduction of Titanium Tetrachloride with Metallic Magnesium

The principal quantity of titanium produced in the world is produced by the magnesium thermal reduction of titanium tetrachloride by the Kroll reaction:



$$\Delta H_{298} = -121,600 + 10,000 \text{ calories}$$

the values of the equilibrium constant (500 degrees Kelvin --  $10^{4.3}$  at  $-2$ , 1000 degrees Kelvin --  $10^{1.7}$  at  $-1$ , and 1300 degrees Kelvin --  $10^4$  at  $-1$ ) indicate a good completion of this process at the temperatures of the technological process.

Magnesium produced by electrolysis is used as the reducing agent.

The titanium is produced in apparatuses called reactors. The design of reactors varies and is determined by the experience in the production of titanium at this or that enterprise.

V. Kroll published an overall view of a 650-kilogram reactor which is being used in a plant in Henderson [GENDERSON/43].

The literature indicates that 1300-kilogram reactors are being created which operate on a batch process. Studies on developing continuous methods for producing titanium are being made on a large scale in different countries, but they have not as yet left the stage of pilot plant projects. The creation of large capacity units for reducing, and the development of continuous methods for producing titanium require painstaking study of the mechanism of the processes of reduction and the formation of sponge titanium under conditions close to production conditions as well as apparatus of the plant type.

A study of the processes of reduction and of formation of sponge titanium was made in the Institute of Metallurgy of the Academy of Sciences, USSR under the leadership of the authors, Engineer S. V. Ogurtsov,

and Candidate of Technical Sciences A. V. Revekin for the case of reduction with sodium.

c. Processes for Reduction and Formation of Sponge Titanium

The reduction of titanium tetrachloride with magnesium is a complicated physico-chemical process whose duration is determined by both physical and chemical phenomena.

An idea was gained of the mechanism of the processes which took place in the reactor as a result of the study of the kinetics of the reaction of reduction of titanium tetrachloride with magnesium, the distribution of the reaction products, the structure of the sponge titanium tetrachloride is fed into the processes of reduction and formation of sponge titanium, and others.

These ideas were formed independently of each other by research workers in different nations.

The works on the study of the mechanism of the processes of the reduction and the formation of sponge titanium underestimate the role of the interaction of the reagents in the gaseous phase, and some of the research workers [4] simply denied the practical possibility of the development of such an interaction. One of the things which confirmed this was the fact that sponge titanium did not grow around metallic pins which had been welded perpendicular to the top of the crucible and did not come in contact with the molten magnesium. Complete reduction of the reaction to the interaction of the tetrachloride with magnesium over the entire surface of the sponge titanium was inaccurate [7]. The causes of the dendritic structure of sponge titanium and the predominant development of the reaction on its surface; the effect of physical phenomena on the mechanism of the process, namely the liquid magnesium chloride, which flows down in the bottom of the reactor over the surface of the titanium sponge; the effect of the delivery of titanium tetrachloride on the formation of the profile of the growing sponge and on the velocity of the process, and so forth likewise have not been explained. Ideas of the mechanism of the processes of reduction and formation of sponge titanium were not linked with the management of the processes in industrial reactors, data on the kinetics of the very process of the reduction of titanium tetrachloride with magnesium, et cetera. Some general ideas concerning the character of the development of the processes of the reduction and the formation of sponge titanium have been gained on the basis of studies made recently both in the USSR as well as abroad.

Prior to the influx of titanium tetrachloride, the molten magnesium does not wet the steel walls of the vessel and has a convex meniscus. In the first instant after the influx of the titanium tetrachloride, the reaction takes place both in the gaseous phase as well as on the surface of the molten magnesium. The character of the interaction of the titanium tetrachloride with the magnesium is shown in Figure 6, which gives a general "kinetic photograph" of the course of the reduc-

tion process. The ordinates in this graph show the degree of development of the interaction of the titanium tetrachloride with vaporized magnesium ( $P_{Mg}$ ) and the evaporation  $TiCl_4$  with magnesium and the rate of the reduction of vaporized  $TiCl_4$  at different stages of the process and at different temperatures. The interaction of  $TiCl_4$  with vaporized magnesium, particularly in the initial period, develops on the walls of the vessel.

The walls are covered with a thin coating of metallic titanium or its lower chlorides and the surface of the molten magnesium with magnesium chloride which isolates the tetrachloride from further direct interaction with the surface of molten magnesium. At the same time, the titanium formed in the first instant settles on the bottom of the vessel and is revealed after separation in the form of a powder or lightly-encrusted films on the bottom of the vessel. The presence of a thin coating of titanium on the walls and the magnesium chloride change the relationship of the surface forces, and the magnesium meniscus under the layer of magnesium chloride changes to a concave form, the magnesium begins to wet the walls of the vessel which are covered with a thin coating of titanium or its chlorides, and climbs up on them. Since the surface of the magnesium is covered over with magnesium chloride, the reaction is developed only where the magnesium climbs over its surface.

Thus, the reaction develops on the walls of the vessel. At the same time, the lateral surface of the sponge which is beginning to grow is very small and is wholly covered by flowing magnesium chloride; but the nidus the reaction is supported only on the crest of the sponge, therefore it is developed in a vertical direction.

The magnesium can move out to the surface only where sections of the surface are freed of magnesium chloride. Such sections are "active" and can grow with the development of "titanium dendrite" which goes past the level of the magnesium chloride flowing over the lateral surface of the reaction mass.

Figure 7 shows the development of dendrites of titanium in the initial period of the reduction of titanium tetrachloride with magnesium. It is obvious that the thinner the layer of magnesium chloride flowing over the lateral surface of the sponge, the greater the number of these sections, and vice versa, the greater the rate of delivery of titanium tetrachloride, the less the outflow of magnesium to the lateral surface and the higher the sponge titanium will climb up the vessel walls. At a certain rate of delivery of titanium tetrachloride, the sponge can reach some definite height where its covering of magnesium chloride is decreased to such an extent that sections of its lateral surface begin to be exposed, magnesium can flow out to these sections, and the sponge titanium begins to grow in a horizontal direction. Such a growth to the limit leads to the formation of the so-called porous "bridge". Consequently, the character of the development of the sponge is determined by the rate of delivery of the tetrachloride can be used to control the formation of the profile of the growing sponge,

giving it a primarily upward development with a high rate of development with a high rate of delivery of the titanium tetrachloride, or toward the center of the reactor when it is delivered at a low rate. By the time that 30 - 40 per cent of the titanium tetrachloride is used, the magnesium has disappeared as an independent layer in the reactor and has gone into the pores of the developing sponge titanium.

At the temperatures prevailing in the process, the reaction mass does not possess sufficient mechanical strength, and the size of the particles of titanium is insignificant, on the order of 500 Angstrom units.

The particles of titanium covered with magnesium are held in a conglomerate by surface forces; in the absence of magnesium the titanium is not in equilibrium with the magnesium chloride and, being located in it, corrodes energetically and forms a mixture of titanium dichloride and magnesium.

A study of the kinetics of the reduction of titanium tetrachloride with magnesium through processing the curves (Figure 6), which were obtained at different temperatures and rates of delivery of titanium tetrachloride, showed that this reaction has an autocatalytic character -- the apparent value of the energy of activation was decreased with an increase in the rate of delivery of titanium tetrachloride (Figure 8). Metallic particles (nuclei) of titanium, on whose surfaces occurred the interaction of titanium tetrachloride with magnesium, which determined the dendritic character of the structure of the sponge titanium, served as the catalyzer of this reaction.

#### d. An Analytical Interpretation of the Processes which Take Place in the Reactor

If one makes a time study of the process of the reduction of titanium in the reactor and constructs a curve of the dependence of the specific pressure of the titanium tetrachloride ( $p$ ) on the rate of its delivery ( $v$ ), then it can be expressed by the simple exponential function

$$p = Av^{-n}$$

where  $p$  - is the specific pressure (millimeters of mercury per kilogram of  $TiCl_4$ ),  $v$  - is the rate of delivery of  $TiCl_4$  in grams per minute,  $A$  and  $n$  - are constants in the equation. This relationship of processes which take place in reducing apparatuses of different capacity is presented graphically in Figure 9. The values of the constants of the equation can be obtained by changing the form of equation (22).

The curve of Figure 9 is a generalized characterization of the processes. The section of the curve where large deliveries of titanium tetrachloride cause its lowest specific pressures in the reactor should be regarded as a region of maximum development of the chemical process where the titanium tetrachloride interacts with the magnesium without increasing the pressure in the reactor. If the process of

reduction in the reactor were to be carried out with delivery of titanium tetrachloride at the rate corresponding to this region of the characteristic curve, then it would be possible to obtain maximum productivity of the reducing apparatus. On the other hand, the section of the curves where high pressures in the reactor correspond to low delivery of titanium tetrachloride is linked with the region of the development of essentially physical phenomena -- evaporation of titanium tetrachloride. If the process in the vessel takes place in this region of values of the rate of delivery of titanium tetrachloride, then only the process of evaporation of liquid titanium tetrachloride is actually going on in the vessel.

The rates of the technological processes taking place in apparatus of different capacity occupy an intermediate position on the characteristic curve between the two limiting regions possible for the course of the processes. The degree of approximation of the technological processes to conditions close to a process which would take place entirely in the region of the maximum possible productivity of the apparatus could be evaluated by the index of the IRS (ispol'zovanie reaktsionnoy sposobnosti - utilization of the reaction capacity). This index is a unique kinetic coefficient of the "efficiency" of the reactor.

The IRS index and the duration of the processes are linearly dependent.

Thus, the closer the conditions of carrying on the process approximate flat portion of the curve (Figure 9), the less the duration of the processes and, consequently, the greater the productivity of the reducing apparatuses.

#### e. The reduction of Titanium Tetrachloride with Metallic Sodium

At present about 30 per cent of the titanium and the principal quantity of the output of this metal in England is produced by the sodium thermal method.

Titanium produced by the sodium method was first obtained in 1887 [45], but it contained a significant amount of admixtures. Malleable titanium was obtained by reducing titanium tetrachloride with sodium in 1910.

In the Soviet Union research on the reduction of titanium tetrachloride was started in the GIREDMET. The technological principles of this process were worked out in the VAMI and in the Institute of Metallurgy of the Academy of Sciences, USSR.

The use of metallic sodium as a reducing agent has a number of advantages over magnesium, including: a) the low temperature at which the process takes place; b) the possibility of delivering the sodium in molten state to the reactor due to its low melting point; c) the insignificant solubility of oxides and gases in it (solubility of oxygen at 98-150 degrees -- 0.005 per cent); d) more complete utilization of sodium in reduction processes 95-99 per cent as compared



with the utilization of magnesium -- 80 - 85 per cent, which is explained by a different mechanism for forming sponge titanium; 9) nonhygroscopic sodium chloride is formed in the process of reducing  $\text{TiCl}_4$ , which permits applying the hydrometallurgical limit of the reaction mass without noticeable deterioration of the quality of the metal. The latter advantage is basic and decisive since it opens great possibilities for utilizing sodium as a reducing agent.

On comparing the cost of sodium and magnesium and the expenditure of electric power for producing them, it follows, however, that the stoichiometric expenditure of sodium is twice the expenditure of magnesium. If one takes into consideration the technological coefficients and compares the sodium and the magnesium thermal processes, then when sodium is used, the cost of the reducing agent per kilogram of titanium produced is lowered by 19 per cent [45]. This decrease in cost cannot play a decisive role at the present stage of the development of the metallurgy of titanium inasmuch as titanium still has very high production costs. The preferential use of this or that reducing agent will be determined by local conditions and general technological practices.

With the growing scale of the titanium industry, it is essential to bear in mind that magnesium can be used as a structural material while sodium can be of importance only as a metallic reducing agent.

A study of the processes of reducing and forming sponge titanium by the sodium thermal method was made in the Institute of Metallurgy of the Academy of Sciences, USSR.

This research showed that the reduction of titanium tetrachloride with sodium begins at 160 degrees (when reactors with agitators are used), but intensive reduction takes place under these conditions at 270 - 320 degrees.

In reactors without agitators, the process of reduction begins at 450 degrees, but stops soon as the sodium chloride which forms interferes with the further progress of the process.

Intensive reduction of titanium tetrachloride with sodium should start at 590 - 550 degrees, and, due to the exothermal nature of the process, should bring the temperature up to the desired limit. Metallic titanium can be produced at temperatures markedly lower than the melting point of sodium chloride (801 degrees), in the temperature range 801 - 880 degrees (the boiling point of metallic sodium) and higher than the boiling point of sodium (temperatures about 880 degrees) by reducing the titanium tetrachloride in the gaseous phase.

At low temperatures, metallic titanium is obtained in the form of a fine powder which, due to its high activity, is readily oxidized in the air.

The mechanism of the processes of reducing and forming sponge titanium in sodium thermal reactors is somewhat different from that in the reduction of titanium tetrachloride with magnesium. This mechanism varies according to the conditions under which the process takes place, when the sodium chloride is formed in solid and liquid form.

The process of reducing  $TiCl_4$  begins at the surface of the molten sodium and, forming a reaction mass, grows higher than the original level of the liquid sodium.

An empty space forms under the reaction mass which is filled with sodium vapor; this space increases as it is used. At the same time, the lower edge of the reaction mass remains at the same level relative to the bottom of the vessel, independent of the utilization of the titanium tetrachloride.

The upper part of the reaction mass is a solid porous mixture of the lower chlorides of titanium and sodium chloride while the lower part is metallic titanium and sodium chloride. Individual particles of titanium formed in the reduction process are not covered by molten sodium; this explains the predominance of fine fractions of titanium [47] obtained by the sodium thermal method.

Size of particles (mesh)	-8	-8	-10	-10	-14	-14	-20	-20	-28	-28	-35	-35	-48
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Per Cent	6.03	7.43	10.08	15.32	8.42	12.20	11.14
Per Cent	6.03	13.46	23.54	38.86	47.28	59.48	70.62

Size of Particles (mesh)	-48	-65	-65	-100	-100	-150	-150	-200	-200	-325	-325
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Per Cent	8.95	8.22	5.57	3.05	2.65	0.93
Per Cent	79.57	87.79	93.36	96.41	99.06	99.99

The process of reducing titanium tetrachloride in the range of temperatures up to 801 degrees begins at the surface of the molten sodium. These first particles of titanium sink into the molten sodium. The surface of the liquid sodium is covered with solid products of the reaction, sodium chloride and titanium, which form the reaction mass.

Vaporized sodium diffuses into the reaction zone through the porous layer of solid products of the reaction which reduces the lower chlorides of titanium formed on the surface of the reaction mass as a result of the interaction of titanium tetrachloride with metallic titanium. After the sodium is distilled off at temperatures at which the titanium particles sinter, the titanium is easily recovered in the form of a very fine powder. This mechanism of the processes of the reduction and formation of sponge titanium in a sodium thermal reactor is determined by the low fusion point of sodium, its high vapor pressures at the temperatures prevailing in the process and the formation of solid reaction products -- titanium and sodium chloride.

At temperatures about 801 degrees, when the sodium chloride is in a liquid state, the formation of sponge titanium is developed on the sides of the vessel and in many ways reminds one of the reduction of titanium tetrachloride with magnesium.

The size of the particles of sponge titanium obtained at high temperatures is increased, and the processes of reduction proceed more intensively than those of the reduction of titanium tetrachloride with magnesium.

#### IV. THE PROCESSING OF THE REACTION MASS

Following the process of reduction and the tapping of a considerable portion of the magnesium chloride, the reaction mass has the following composition: titanium 65 - 70 per cent, magnesium chloride 20 - 25 per cent, magnesium 10 - 20 per cent, and some quantity of the lower oxides of titanium. The sponge titanium which is to be sent out for smelting should not contain more than 0.1 per cent magnesium and 0.15 per cent chlorine.

Decreasing the content of these impurities is achieved by means of processing the reaction mass, first by leaching out the magnesium chloride and the magnesium with acidified water. The technological principles of the hydrometallurgical procedure for processing the reaction mass were worked out by the GIREDMET and introduced in the Podol'sk Plant.

Significant losses and deterioration of the properties of metallic titanium take place in the hydrometallurgical processing of the reaction mass in connection with the absorption of oxygen and hydrogen. At one time the industry refused this reason to work with hydrometallurgical processing and changed over to vacuum thermal processing of the reaction mass (separation). The study and the development of the principles of vacuum thermal processing in production apparatuses were carried out by the GIREDMET, The All-Union Scientific Research Institute for New Materials, the VAMI, and other institutes while the mastery of the industrial technology and the development of designs for modern apparatus for vacuum thermal processing were carried out jointly by the Podol'sk Chemico-Metallurgical Plant and the VAMI and other special planning and research institutes. Systematic research was started on the process of separation in the Institute of Metallurgy of the Academy of Sciences, USSR.

A generalization of the results of the research, which was conducted by Candidates of Technical Sciences, A. V. Revyakin and V. S. Mirochnikov showed that the quality of metallurgical titanium is essentially determined by painstaking work in vacuum thermal processing.

The reaction mass is exceedingly hygroscopic. Absorption of moisture by the products of reduction takes place as a result of the interaction of magnesium chloride and the lower chlorides of titanium with water. The compounds  $MgCl_2 \cdot nH_2O$ , where  $n$  varies from 1 to 6, hydrochloric acid, and meta-titanic acid are formed as a result of hydrolysis. If special measures are not taken, the oxide compounds will interact with titanium at the temperatures of the vacuum thermal processing, forming solid solutions of oxygen in it. This lowers the mechanical properties of the titanium and its quality.

In order to limit the hydrolysis of the chlorides, the reaction mass is subjected to processing on specially-designed machine tools under conditions of reduced moisture prior to being sent out for separation. This low moisture is created in special "drying chambers". Some plants accomplish separation of the reaction mass in a monolithic form. It was established by studies of the separation of the reduction products, both in chip and in monolithic form, that several periods can be distinguished in the process of vacuum thermal processing, depending upon the temperature, which are characterized by the progress of certain processes: 0 - 350 degrees -- the release of moisture; 350 - 600 degrees -- the elimination (essentially) of hydrogen; 650 - 900 degrees -- the elimination of magnesium and magnesium chloride. The processes are completed with soaking at a temperature of 900 - 950 degrees to remove the remaining quantities of magnesium, magnesium chloride, and hydrogen.

Changes in the weight of the reaction mass are given in Table 5.

TABLE 5

Processes	Loss of Weight (Per Cent)		
	0 - 390°	354 - 600°	601 - 900°
1	3.32	1.25	89.4
2	6.40	3.26	88.6
3	5.54	1.63	86.9
4	3.94	3.28	89.2

The first period is distinguished by a small decrease in the vacuum due to an abundance of gas (vapor). After 200 degrees, the change in weight is insignificant, and the vacuum remains quite low. Qualitative analyses were made of the discharge gases, for which purpose two vacuum tubes separated by a trap cooled with liquid nitrogen were installed on the exhaust line. The first tube fixed the total pressure ( $P_1$ ) and the second -- only the hydrogen pressure ( $P_2$ ).

The results of the studies of the separation of the reaction mass (Figure 10) showed that up to 350 degrees only moisture was discharged and hydrogen was not discovered in the gaseous phase. Above 350 degrees, the appearance of hydrogen began to be observed in the gaseous phase, and the amount of hydrogen increased sharply with higher temperatures.

Above 480 degrees (the second period of separation), the gaseous phase consisted almost entirely of hydrogen. If we compare the analysis of the gaseous phase and the loss of weight of the reaction mass ( $Q_{rp}$ ), then in the region of the discharge of moisture the mean loss of weight amounts to more than 4.3 per cent, while at the temperatures of the discharge of hydrogen the loss of weight does not exceed 2.3 per cent. A comparison of the figures of the loss of weight and an

analysis of the gases discharged during separation permit one to conclude that in the first period, up to 350 degrees, the titanium is not oxidized by the moisture, and above 480 degrees (the second period of separation), the moisture is obviously completely decomposed by the titanium, forming titanium dioxide and hydrogen. Since only the monohydrate of magnesium chloride ( $\text{MgCl}_2 \cdot \text{H}_2\text{O}$ ) can exist at temperatures above 350 degrees, the oxidation of titanium takes place essentially through the decomposition of the monohydrate of magnesium chloride. If one could succeed in completely dehydrating the magnesium chloride at temperatures at which oxidation of the titanium would not take place, then it would be possible to prevent oxidation of the titanium in the process of separation at high temperatures and there would be no need to use special "dry chambers" for processing reaction masses and assembly and disassembly of the reactors. However, the studies showed that even prolonged soaking in vacuum at a temperature of 250 degrees did not succeed in completely removing moisture.

In the second period of separation oxidation of sponge titanium takes place, since hydrogen appears in the gaseous phase at a temperature above 350 degrees -- the product of the decomposition of moisture and the oxidation of titanium.

It is known that the rate of absorption of hydrogen by titanium is proportional to the square root of its pressure; consequently, it is possible that hydrogen is absorbed by sponge titanium in this period, along with the process of oxidation.

Removal of metallic magnesium and magnesium chloride (the third period of separation) begins with heating the reaction mass to temperatures above 600 degrees. Study of the evaporation of metallic magnesium and magnesium chloride has shown distillation of these components in vacuum begins to be intensive at temperatures of 550 - 580, and 650 - 680, respectively.

The dependence of the process of evaporation of these components, both in the pure form as well as from sponge titanium under the same conditions for carrying out the process, upon temperature is about the same; thus, the rate of removal of these admixtures from the reaction mass is determined essentially by the rate of their evaporation which, as is well known, depends upon the intensity of the heating of the sponge titanium in vacuum thermal processing units.

In the third period of separation, along with the process of evaporation of metallic magnesium and magnesium chloride, there is a process of crystallization and formation of sponge titanium, an increase in size of the particles of titanium which appeared in the reduction period, and removal of a certain amount of hydrogen at temperatures around 900 degrees. The process of crystallization and formation of sponge titanium proceeds as the magnesium is being removed, which cements the particles of titanium and, along with magnesium chloride, forms the conglomerate that is called the reaction mass.

Diagrams of separation apparatuses with upper and lower condensers were shown at the All-Union Industrial Exposition.

The duration of the processes in apparatuses with upper and lower condensers with their reaction pots in upright and inverted positions was about the same. The studies which were made showed that in apparatuses with a lower condenser the rate of removal of the principal part (97 - 98 per cent) of the admixtures was markedly higher than that of apparatuses with an upper condenser. The consumption of electric power per unit weight of titanium was somewhat lower for separation in inverted pots in apparatuses with a lower condenser than it was for apparatuses with an upper condenser.

Studies showed that the temperature for carrying on the process had a great effect on its duration. It was determined experimentally that the same degree of completeness of removal of magnesium was achieved in half the time at 925 degrees that it took at 900 degrees.

The process of separation is completed by soaking at 900 - 950 degrees, the duration of which has a great influence on the total duration of the separation process. The principle amount of electric power is consumed in removing the very small remaining quantities of admixtures.

Research on the effect of the vacuum on the quality of the sponge titanium established that high vacuum does not noticeably improve the quality of the sponge titanium and does not increase the rate of separation which is essentially determined by the diffusion of the metallic magnesium and magnesium chloride contained in the sponge titanium, and not by the rate of diffusion of the vapors of the substances in the condensation zone [48].

Processing the reaction mass by the vacuum thermal method is expensive and requires much electric power; therefore, research with the objective of perfecting this process is an urgent and vital task for the appropriate institutes and enterprises. Studies conducted recently have established that as a result of the application of certain improvements in the hydrometallurgical method for processing the reaction mass, sponge titanium can be produced which is equal to the sponge obtained from vacuum thermal processing in respect to the amount of gases contained in it.

A number of suggestions have appeared in the literature on simplifying the process of separation. It is known that in vacuum thermal processing, the greatest amount of time is used in distilling off the last remaining amounts of admixtures; therefore, a two-stage process is suggested: the first stage -- vacuum distillation of a large portion of the metallic magnesium and magnesium chloride, which does not involve large expenditures, and the second stage -- leaching out the sponge titanium which is almost free of magnesium chloride. Such a combination would require only a third of the existing machinery for separation and a small amount of machinery for acid processing.

The suggestion of combining the processes of reduction and separation in a single apparatus, a model of which was shown in the All-Union Industrial Exposition, is of interest. Such a process would al-



low one to abandon the "dry chambers"; and the distillation of admixtures would proceed more rapidly in it on account of better heat transfer and retention of the majority of the capillaries in the sponge undamaged, through which the magnesium climbs up into the zone of interaction, which would facilitate liberation of the admixtures contained in the reaction mass.

## RESEARCH IN THE FIELD OF TITANIUM DIOXIDE RECOVERY FROM COMPLEX TITANIUM-BEARING RAW MATERIAL

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At present the world production of titanium dioxide is about 600,000 - 700,000 tons per year, of which the United States accounts for about 450,000 tons. Two thirds of the entire amount of titanium dioxide produced is sent to the pigment industry and one third to other branches of production (fillers for rubber, paper, silk, etc cetera). In this connection research in the field of production of titanium dioxide has seen extensive development.

The principal tasks faced by research workers were essentially the search for more economical methods for processing the raw material, new types of pigments, and also included studies of their properties. The literature available on these problems is represented by patents and different descriptions of technological processes. Therefore, many problems in regard to phase equilibrium, solubility of titanium compounds, and the study of their physico-chemical properties have received inadequate elucidation.

Up to the present, the production of titanium dioxide in the USSR has been on a small scale. Therefore, the research done on the oxygen compounds of titanium has not acquired its deserved development. However, a number of circumstances, also the character of the titanium raw material in the USSR, have facilitated the development of physico-chemical research on the production of double sulfate salts of titanium with cations of monovalent metals.

The discovery in 1934 of industrial deposits of loparite (a complex titanoniobate of rare earths) and knopite (calcium titanate which contains rare earths and niobium) in the Kola Peninsula attracted the attention of many research workers to it. A great amount of work in this direction was done by A. Ye. Fersman [1]. He interested the principal research organizations in the new type of titanium raw material -- the NIILK [Nauchno-issledovatel'skiy institut lakokrasochnoy

promyshlennosti - Scientific research Institute of the Pigment Industry<sup>7</sup>, the GIPKh [Gosudarstvennyy institut prikladnoy khimii - State Institute of Applied Chemistry], and a number of institutes of the Academy of Sciences. Therefore, much research had been done in the USSR, in contrast to the situation abroad, in the field of processing complex titanium raw material, in particular, the ore from Kola Peninsula, and comparatively little had been done on the processing of ilmenite.

Studies have been started in recent years on the recovery of titanium dioxide from slags whose production was organized on ilmenite concentrates from different ore deposits and also on titanomagnetite.

The production of titanium dioxide from ilmenite suffers from a number of shortcomings: the large outlays for the sulfuric acid which goes into the formation of iron sulfate; prior to hydrolysis of the titanium sulfate to free the titanium solutions from iron, it is necessary to freeze the latter (in the form of its heptahydrated sulfate), after which the titanium solutions are concentrated by evaporation. Thus, the production of large amounts of titanium dioxide will bring about the accumulation of heptahydrate iron sulfate, for which no market is assured. It is possible to obtain 3.1 tons of this product for every ton of titanium dioxide. Therefore, the direct processing of ilmenite concentrate with sulfuric in the amount of 1.1 tons per ton of titanium dioxide in addition to the iron contained in the concentrate. It should be noted that 2.58 tons of concentrate ( $\text{TiO}_2$  content of 43 per cent) and 4.10 tons of sulfuric acid (monohydrate) are consumed per ton of titanium dioxide produced. The waste sulfuric acid, containing 310 grams per liter of  $\text{H}_2\text{SO}_4$  and 41 grams per liter of iron, yields poorly to regeneration.

Studies made by M. L. Borodina and co-workers (GIPI-4 [Transliterated, meaning of the abbreviation is not known<sup>7</sup>]) in the field of processing titanium slags and extracting titanium dioxide from them permits one to compare the methods of producing titanium dioxide from titanium slags and ilmenite concentrate. The solutions obtained after the titanium sulfate is leached out are such that there is no further necessity for concentrating them prior to hydrolysis. The hydrolyzed sulfuric acid obtained contains 324 grams per liter of  $\text{H}_2\text{SO}_4$  and only 9.0 grams of iron, thus yields more readily to regeneration. The complete possibility of processing slag containing 42 - 55 per cent  $\text{TiO}_2$  for titanium dioxide has been established in addition to the possibility of processing slag with a high titanium dioxide content. The research has opened new prospects for the use of slags from the electric smelting of titanomagnetite. The latter is very important since the Soviet Union has enormous reserves of titanomagnetite ores whose complex processing would permit the recovery of iron, titanium, and other accessory elements.

Thus, the problem of obtaining titanium dioxide from slags is not causing any fundamental objections from the technological point of

view at the present time. The economy of the use of the raw material is the basic problem which must be solved when selecting the raw material for titanium dioxide production. The variety of economic and geographical conditions and particularly the complexity of the raw material in the Soviet Union permit the finding of solutions in every case which correspond to the most profitable methods for producing titanium dioxide. Therefore, research has been developed in the field of complex utilization of titanium-containing raw material.

Loparite, perovskite, and titanite concentrates constitute the most complicated types of complex titanium raw material in the Soviet Union. The study of methods for processing them was undertaken by the State Institute of Applied Chemistry, the Institutes of the pigment industry, of general and inorganic chemistry, of rare metals, the VIMS [Vesoyuznyy institut mineral'nogo syr'ya - All-Union Institute of Mineral Raw Materials], the Institute of Metallurgy of the Academy of Sciences, the Urals and Kola Branches of the Academy of Sciences (UFAN and KOLFAN), and the Planning and Research Institute (GPI-4).

As a result of the work that has been done, there are at present a number of methods which insure not only the extraction of titanium dioxide from these concentrates, but also the recovery of the remaining valuable constituents. A number of methods which make use of sulfuric, nitric, and hydrochloric acids have been suggested for the processing of the concentrates. However, the difficulties caused by the use of acid methods of decomposition have not been completely solved as yet.

P. S. Kindyakov (IONKh) used in sulfuric acid first to decompose loparite concentrate [2]. His idea -- to make use of the difference in solubility of niobium sulfate and titanium sulfate -- turned out to be impossible in practice as it was impossible to obtain niobium concentrate free of titanium and titanium concentrate free of niobium.

Subsequently O. M. Gvozdeva (GIREDMET) suggested the decomposition of loparite concentrate with 70 - 80 per cent sulfuric acid at 170 - 180 degrees [3]. After the sinter was leached out with water, a certain amount of titanyl sulfate was added to the solution as seed which caused the entire mass of titanium to crystallize in the form of titanyl sulfate. This method turned out to be quite ineffectual, even though the separation of titanium and niobium proceeded considerably better with this method than it did when I. S. Kindyakov's method was used.

In 1941 V. S. Syrokomskiy and co-workers suggested a method for differential hydrolysis based on strict regulation of the pH of the solution, in which part of the titanium is in the trivalent state [4]. Subsequently, A. I. Ivanov (VIMS) continued research in this direction, applying these ideas to the processing of perovskite concentrates. Due to the low coefficient of recovery of niobium and the large admixture of titanium, however, these methods were not extensively developed and acquired no practical application.

Research was started at almost the same time in the KolFAN and UFAN on the processing of loparite concentrate by fusing it in a bisulphate alloy. Ya. G. Goroshchenko (KolFAN) [5] suggested fusing loparite concentrate in a mixture of molten ammonium sulfate and sulfuric acid [5], and N. V. Demenev and co-workers (UFAN) in a mixture of potassium sulfate and sulfuric acid [6]. The research done in the KolFAN was more complete and included large-scale experiments. As a result of this research, three methods were suggested for processing complex loparite concentrate with 90 - 92 per cent recovery of titanium dioxide, 75 - 80 per cent recovery of niobium with tantalum (with an 80 per cent niobium product), and up to 85 per cent recovery of rare earths. At the same time, the problem of regenerating spent sulfuric acid, which was to be reclaimed in the form of ammonium sulfate, was solved. In the first case the method repeated the general outline of O. M. Gvozdeva's method. It was improved by Ya. G. Goroshchenko [5] by selecting the conditions for fusing the concentrates and salting out the titanyl sulfate. In the second case, the conditions for fusing the concentrate were selected in such a manner as to result in forming double ammonium sulfate salts with titanium --  $(\text{NH}_4)_2\text{Ti}(\text{SO}_4)_3$  and niobium  $\text{NH}_4\text{NbO}(\text{SO}_4)_2$ . Finally, double ammonium sulfate salts with titanyl  $(\text{NH}_4)_2\text{TiO}(\text{SO}_4)_2 \cdot \text{H}_2\text{O}$  and niobyl  $\text{NH}_4\text{NbO}(\text{SO}_4)_2$  can be obtained under certain conditions of fusing. The last two salts possess different solubility in aqueous solutions of sulfuric acid and ammonium sulfate, which permits their very successful production in a pure form. After the double salts of titanium and the ammonium sulfate are hydrolyzed, titanium dioxide can be produced which has an admixture of not more than 0.2 - 0.4 per cent of niobium pentoxide and other impurities within limits of 0.3 per cent. The UFAN method [6] which is based on the use of potassium sulfate also permits obtaining double potassium sulfate is regenerated in the form of metatitanic acid after the titanium is separated out by hydrolysis.

It follows from the foregoing that effective acid methods have been suggested for complex processing of loparite concentrates with which it is possible to recover all useful constituents of the concentrates. The practical use of these methods will depend on technical and economic conditions; and the possibility of applying them should be settled separately for each case.

Methods were developed at the same time for acid processing of perovskite concentrates. It should be noted that perovskite raw material, like the loparite, has no analogs in world practice of exploitation of titanium ores and thus the technology for processing them from mining to producing titanium dioxide required new, original methods for processing. The method developed by M. A. Shtern (NIILK) [7] and I. V. Riskin (GPI-4) based on the decomposition of finely-ground perovskite concentrate by sulfuric acid in the presence of a 5 per cent iron powder is the simplest method of processing. The powder is added to prevent hardening of the reacting mass and to reduce part of the titanium to  $\text{Ti}^{3+}$  at the instant it is leached out. After the ti-

tanium sulfate solution is concentrated to 250 grams per liter of  $\text{TiO}_2$ , metatitanic acid is separated out of the boiling solution, by hydrolysis. The hydrolyzed sulfuric acid can be regenerated or, after suitable treatment to render it harmless, sent out to the dump. Metatitanic acid is converted to titanium dioxide by heating. With such a method of processing, perovskite concentrates suffer an irreversible loss of rare earths and niobium [columbium] with tantalum.

Of the methods which permit complex utilization of the valuable constituents of perovskite concentrates, one should mention the method of Ya. G. Goroshchenko (KolFAN) [8]. The author and M. L. Borodina subsequently improved and simplified this method. In one case the method was based on the fusion of perovskite concentrates in a mixture of sulfuric acid and ammonium sulfate; and in the other, on the decomposition of the concentrate by sulfuric acid with small quantities of ammonium sulfate added. In the first method, after separation of the slimes, the titanium was salted out in the form  $(\text{NH}_4)_2\text{SO}_4 \cdot \text{Ti}(\text{SO}_4)_2$  while the niobium remained in solution. Subsequently, the aforementioned double salt was subjected to hydrolysis under conditions analogous to the production of metatitanic acid from titanium sulfate. As a result of further processing the solution obtained after separation of the double sulfates of titanium and ammonium, it is possible to extract the titanium sulfate. As a result of further processing the solution obtained after separation of the double sulfates of titanium and ammonium, it is possible to extract the titanium, to obtain niobium and tantalum concentrates containing up to 60 - 70 per cent of their pentoxides, and also rare earth concentrates. For this purpose, the solution is concentrated by evaporation, then the ammonium ammonium bisulfate and sulfuric acid are almost wholly distilled out at 200 - 220 degrees. In the other method, after the slimes consisting essentially of calcium sulfate, undecomposed minerals and silica are separated, titanium, niobium with tantalum, and about 5 per cent of the rare earths go into solution. The double sulfates of titanium and ammonium are salted out of this solution, then the titanium salt is processed just as it was in the first case. Subsequent processing of the niobium solution is distinguished by the fact that the niobium, tantalum, iron, and titanium and rare earth residues are precipitated by ammonia. Then the hydrate precipitate is dissolved in sulfuric acid and processed as in the first case.

In addition to the sulfate processes for the complex utilization of perovskite, B. N. Melent'yev and co-workers [5] suggested a nitric acid process for processing it. This process makes use of the property of perovskite -- that of decomposing in nitric acid at its boiling point and at higher temperatures. In this case, slimes are formed which consist of metatitanic, and niobic acids, silica, and undecomposed minerals. The calcium, and magnesium, rare earths, and other constituents of the concentrates which are soluble in the

acid go into the nitric acid solution. The slimes can be processed with the sulfate method, producing titanium dioxide and niobium concentrates. Concentrates of the rare earths are separated out of the nitric acid solution in the form of hydrates, while the calcium nitrate in solution is converted into ammonium nitrate.

Such a method of processing ensures high coefficients of recovery of the constituents of the concentrates, also saving the nitric acid which, after doing its work in decomposing the perovskite, is recovered in the form of a commercial product (calcium nitrate or ammonium nitrate).

A metallurgical process for working with perovskite concentrates suggested by K. Kh. Tagirov and co-workers (IMET) of the Academy of Sciences, USSR, permits the use of titanomagnetite, too. The process is based on the electric smelting of a mixed perovskite-titanomagnetite concentrate, transferring the titanium into slag and the niobium into cast iron. Subsequent treatment of the cast iron in a converter yields steel and niobium slag, from which the niobium can be recovered by acid processing. Titanium dioxide is recovered from the titanium slag with sulfuric acid.

Several methods were suggested for producing titanium dioxide and pigments of various colors in the processing of titanite concentrates.

Titanite concentrates can be processed into titanium dioxide with the aid of sulfuric acid. However, the large consumption of sulfuric acid and the large quantity of tailings formed (gypsum) makes this type of raw material quite unprofitable for the producing of titanium. Still, it can be used in the production of the so-called SILITAN [Translator's note: English equivalent of the word is not known, possibly silitane] (a mixture of the dioxides of titanium and silica), which has good pigment properties and is capable of being colored with organic dyes. The production of one ton of SILITAN requires 2 tons of titanite concentrates and 3 tons of hydrochloric acid.

It follows from the foregoing that at present methods have been worked out which permit the rational utilization of complex titanium raw materials in the Soviet Union.

Important work has been done in the study of the properties of titanium sulfate compounds.

D. L. Motov (KOLFAN) made a detailed study of the  $(\text{NH}_4)_2\text{SO}_4$  --  $\text{TiO}_2$  --  $\text{H}_2\text{SO}_4$  --  $\text{H}_2\text{O}$  system at four isotherms: 0, 20, 40, and 80 degrees. During this research, regions of stability were established for the compounds  $(\text{NH}_4)_2\text{TiO}(\text{SO}_4)_2 \cdot \text{H}_2\text{O}$ ,  $(\text{NH}_4)\text{TiO}(\text{SO}_4)$ , and  $(\text{NH}_4)_2\text{Ti}(\text{SO}_4)_3$ . The temperature points for changes in phase were discovered. The conditions for forming these salts were established.

Ya. G. Goroshchenko (KOLFAN) [5] was first to direct attention to their existence. He established that salt of the composition  $(\text{NH}_4)_2\text{TiO}(\text{SO}_4)_2 \cdot \text{H}_2\text{O}$  was stable at temperatures below 40 degrees, but its anhydrous modification  $(\text{NH}_4)_2\text{TiO}(\text{SO}_4)_2$  was stable at temperatures



above 40 degrees. While studying the properties of these salts he established that the salt  $(\text{NH}_4)_2\text{Ti}(\text{SO}_4)_2 \cdot \text{H}_2\text{O}$  is represented by isotropic tetrahedrons with an index of refraction of 1.580. The anhydrous salt, however, has anisotropic  $\alpha$  and  $\beta$  modifications, of which the  $\alpha$  modification possesses right extinction and indexes of refraction: along Ng - 1.707 and along Np - 1.600.

The  $\beta$  modification is represented by tetrahedrons with a large angle between the optical axes and the index of refraction of N = 1.629. Salt of the composition  $(\text{NH}_4)_3\text{Ti}(\text{SO}_4)_3$  separated out of the ammonium sulfate alloy, or specially synthesized, decomposes under the action of atmospheric moisture and is converted either into anhydrous titanyl sulfate. When heated, the salt begins to decompose at about 400 degrees. Under the microscope the crystals of the salt have the appearance of hexagons. The extinction is right, Ng = 1.754. and Np = 1.684.

Ya. G. Goroshchenko [10] also obtained the following double salts of titanium:  $\text{CaTi}(\text{SO}_4)_3$ , which yields isomorphous substitutions with  $(\text{NH}_4)_2\text{Ti}(\text{SO}_4)_3$ . The salt  $\text{TR}(\text{SO}_4)_3 \cdot 4\text{Ti}(\text{SO}_4)_2$  crystallizes at temperatures above 200 degrees in ammonium sulfate alloys in case rare earths are present. However, formation of the salt of the composition  $\text{R}_2(\text{SO}_4)_3 \cdot 2\text{Ti}(\text{SO}_4)_2$  where R --  $\text{TR}^{3+}$ ,  $\text{F}^{3+}$ ,  $\text{NH}_4^+$ , is also possible.

The salt  $2(\text{NH}_4)_2\text{SO}_4 \cdot 3\text{TiOSO}_4 \cdot \text{Ti}(\text{SO}_4)_2$  is formed under the same conditions. By determining the number of transfers for  $(\text{NH}_4)_2\text{Ti}(\text{SO}_4)_3$ , Ya. G. Goroshchenko established that the titanium enters the complex ion  $[\text{TiO}(\text{SO}_4)_2]^{2-}$ .

N. V. Demenev and co-workers [6] studies the conditions for crystallization of the salt  $2\text{K}_2\text{SO}_4 \cdot 2\text{TiOSO}_4 \cdot 5\text{H}_2\text{O}$  which is formed in a solution containing 250 - 400 grams per liter of  $\text{H}_2\text{SO}_4$  and 20 - 30 per cent of  $\text{K}_2\text{SO}_4$ . They also established the form of the crystals with the aid of an electron microscope and electron diffraction studies.

B. Ye. Boguslavskaya and O. M. Ottomanovskaya [11] established in 50 - 60 per cent sulfuric acid, hydrated metatitanic acid crystallizes into  $\text{TiOSO}_4 \cdot 2\text{H}_2\text{O}$  at 95 - 100 degrees. The very same compound was obtained by Pamyilov and Khudyskov under almost analogous conditions. However, Ya. G. Goroshchenko states that a salt of the composition described above cannot be crystallized under these conditions. Boguslavskaya synthesized colorless crystals  $\text{Ti}(\text{SO}_4)_2 \cdot 4\text{H}_2\text{O}$  from a sulfuric acid solution containing potassium fluoride. Other oxygen compounds of titanium were studied to a markedly smaller extent. When studying phosphates of titanium, V. I. Spitsin and Ye. A. Ippolitova [12] came to the conclusion that they did not have crystalline structures and were x-ray amorphous. They established that precipitated titanium phosphate loses water during heating and does not yield stable compounds of stoichiometric composition. Titanium phosphate calcined at 1000 degrees corresponds approximately to  $2\text{TiO}_2 \cdot \text{P}_2\text{O}_5$ . Metaphosphates in aqueous solutions do not yield precipitates with titanium salts. Pyrophosphates, even though they do precipitate ti-

tanium, are readily soluble in an excess of the precipitating agent. Therefore, the authors believe that the  $P_2O_5$  is absorbed by the metatitanic acid and the amount of phosphoric acid trapped will depend essentially on the method of preparing the titanium gel.

I. N. Belyayev [13, 14, 15, 16] and co-workers studies diagrams of fusability (up to 1100 degrees) of lead titanate with fluorides, silicates, pyrophosphates, and carbonates of sodium and potassium. An analogy was established between the chemical properties of lead titanate and barium titanate. They did extensive research on diagrams of the fusability of barium titanate with fluorides, silicates, pyro- and metaphosphates, metaborates, chlorides, molybdenates, sulfates, tungstates, and carbonates of the alkali and some divalent metals. It was established in this manner that barium titanate is scarcely soluble in the majority of salt alloys of Na, K, Li, Pb, and Ba. At the same time, regions of separation were determined in the systems under study. The authors established that the structure of barium titanate is an infinite three-dimensional anion of regular or deformed ( $TiO_6$ ) octahedrons with a titanium ion in the center. Barium titanate is capable of forming solid solutions with silicates, borates, and other compounds which are distinguished by a highly polymerized molecular structure.

B. A. Bron and A. K. Podnigin [17] synthesized and studies the properties of aluminotitanate --  $AlTiO_5$ . For it they found a high birefringence  $N_g$  2.06 and  $N_p$  2.025; melting point of 1890 - 10 degrees, specific gravity of 3.681 and a coefficient of thermal expansion of  $8 \times 10^{-8}$  in a temperature range of 0 - 800 degrees. The authors recommend aluminotitanate as a substitute for quartz in many industrial items.

Studies in the field of analysis were made primarily of metallic titanium. The colorimetric reaction with hydrogen peroxide which has been widely used for determining the presence of titanium was studied in detail by A. K. Bobko and A. I. Volkova [18]. They established that two peroxide compounds are formed at a ratio of  $Ti : H_2O = 1 : 1$ . One of them is colored in the interval  $[H^+]$  from 3 to 6 pH, and the other is colorless at pH 5 and higher. The authors consider that the colored peroxide compound of titanium is addition compound, that is, the  $H_2O$  molecule is a coordinate group. On the other hand, the colorless compounds contains  $HO^-$  or  $O^-$  ions as coordinate groups. The calculated dissociation constant for the colored peroxide titanium compound  $K_{TiH_2O_2} 0.9 \times 10^{-4}$ .

A number of methods for isolating and separating iron, molybdenum, vanadium, and titanium with ion-exchange resins was suggested by A. M. Medvedev and L. M. Orlova, G. T. Galdayan [19] used an electrolytic method for reducing titanium in its volumetric analysis. B. M. Peshkova, and Z. A. Gallay, Yu. I. Usatenko and G. Ye. Bekleshova, Z. S. Mukhina [19] and other suggested current measuring or polarographic methods be used for analysis of titanium. Yu. A. Chernikhov and co-workers [19] published their studies on the analysis of

metallic titanium for impurities. Methods for spectral analysis to determine impurities in metallic titanium were suggested by Sh. G. Melamed, N. S. Sventitskiy, K. A. Sukhenko, and others [19].

It follows from the brief survey presented here of the status of research in the oxygen compounds of titanium that research work in this field has not attained its deserved development as yet. Systematic studies have not been made. In connection with the complex loparite and perovskite ores being brought into industrial exploitation, research should be continued on the double and triple systems represented by  $(\text{NH}_4)_2\text{O}$ ,  $\text{K}_2\text{O}$ ,  $\text{Na}_2\text{O}$ ,  $\text{H}_2\text{SO}_4$ ,  $\text{H}_2\text{O}$ ,  $\text{TR}_2\text{O}_4$ ,  $\text{CaO}$ , and  $\text{TiO}_2$ . It is necessary to continue the study of the phosphates of titanium since they can form the basis for organizing the processing of perovskite and loparite ores. A great deal of research should be done to study procedures for processing titanium slags into titanium dioxide. Such research should solve not only technological problems, but also those which are connected with the study of the kinetics of dissolving the slags in different acids, the thermochemistry of their solution, and the formation of new phases.

It is essential to make extensive studies of ways for processing slags which contain 45 - 55 per cent  $\text{TiO}_2$ . In the first instance, it is necessary to study the possibility of concentrating them chemically in order to obtain a product containing not less than 75 per cent  $\text{TiO}_2$ .

In all cases, it is necessary to study the phase composition of the products obtained and the properties of the phase formed.

Carrying out extensive, organized research will make possible the rapid development of the production of titanium dioxide in the Soviet Union.

# FIGURE APPENDIX

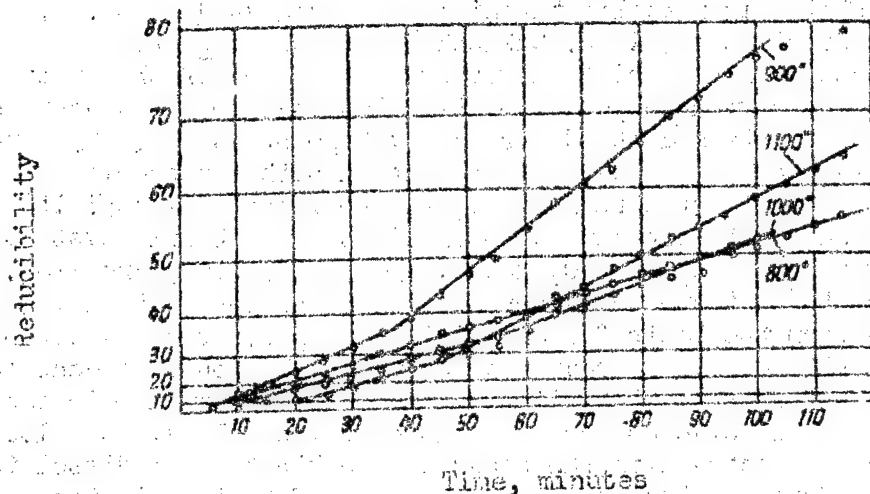


Figure 1. The reducibility of agglomerate prepared by sintering titanomagnetite concentrate in a stream of carbon monoxide. The size of the particles of the agglomerate in 2 - 3 millimeters.

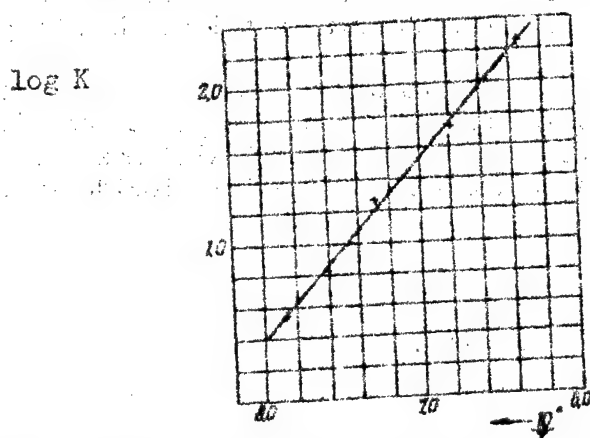


Figure 2. The dependence of the constant of reducibility (K) on the temperature during the reduction of ilmenite by solid carbon.

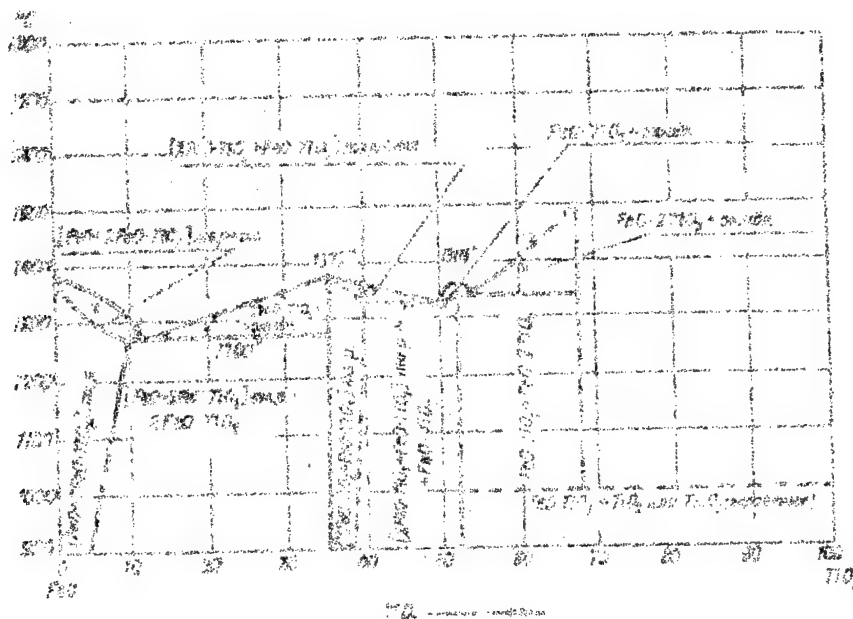
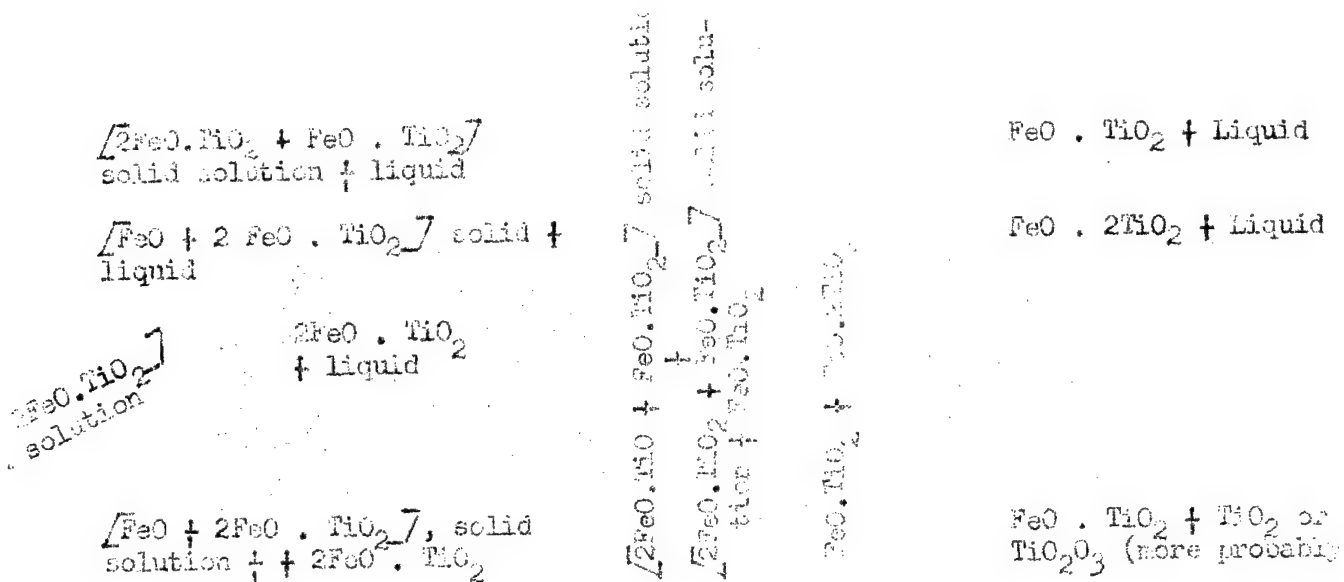


Figure 3. A diagram of the state of  $\text{FeO} - \text{TiO}_2$  according to data of Tsvetkov and Shchepochkina (Academy of sciences, USSR)

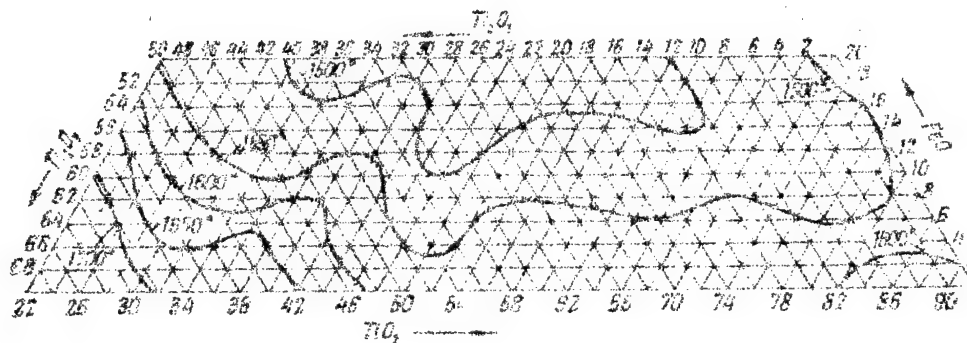


Figure 4. Diagram of the fusibility of the  $\text{TiO}_2 - \text{SiO}_2 - \text{Ti}_2\text{O}_3 - \text{Al}_2\text{O}_3 - \text{Na}_2\text{O} - \text{FeO}$  system with a constant content:  $\text{SiO}_2 - 4$  per cent,  $\text{Al}_2\text{O}_3 - 2$  percent, and  $\text{Na}_2\text{O} - 2$  per cent.

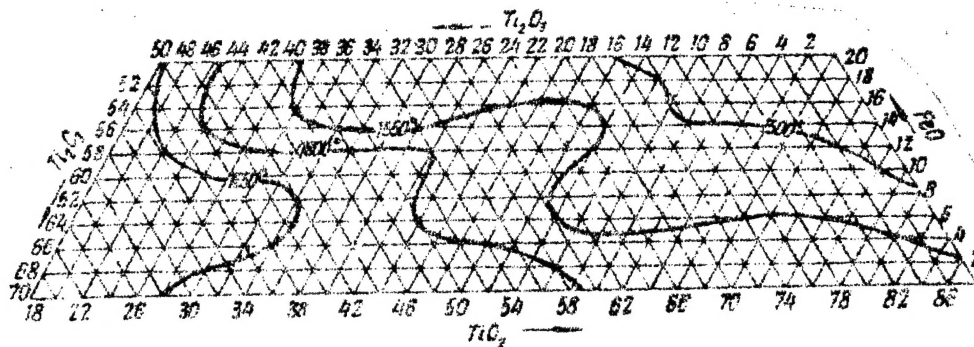


Figure 5. Diagram of the fusability of the  $\text{TiO}_2$  --  $\text{SiO}_2$  --  $\text{Ti}_2\text{O}_3$  --  $\text{Al}_2\text{O}_3$  --  $\text{MgO}$  --  $\text{FeO}$  system with a constant content:  $\text{SiO}_2$  - 34 per cent,  $\text{Al}_2\text{O}_3$  - 2 per cent, and  $\text{MgO}$  - 6 per cent.

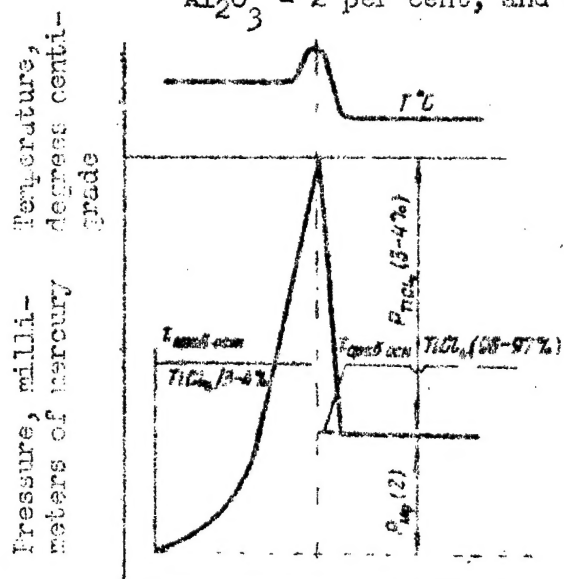


Figure 6. Changes in temperature and pressure in an apparatus for reducing titanium tetrachloride with magnesium. An interpretation of the kinetic curve of the reduction of  $\text{TiCl}_4$  with magnesium.



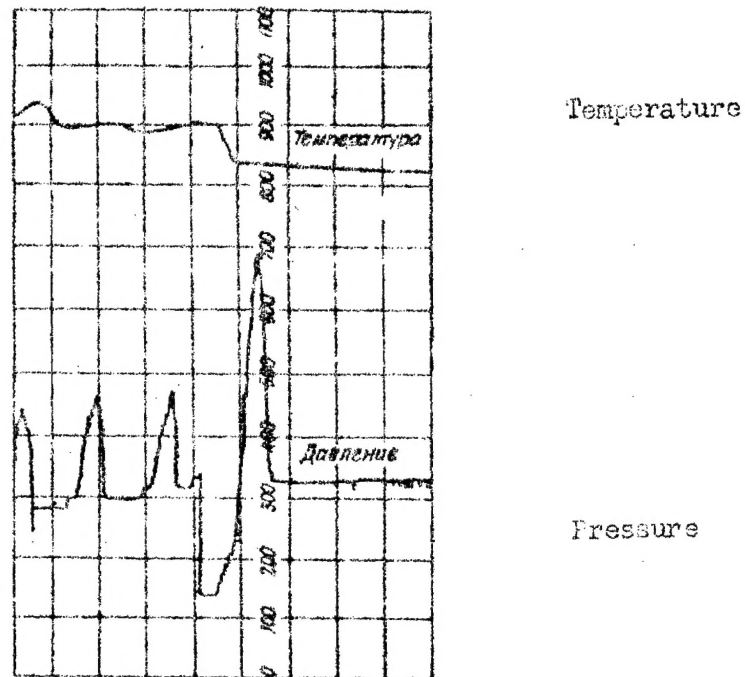


Figure 6. Joint recording of curves of changes in temperature and pressure with an instrument.



Figure 7. Microstructure of sponge titanium. Use of  $\text{TiCl}_4$  - 20 per cent. (Light sections - dendrites of titanium; dark sections - magnesium)

$\log v$

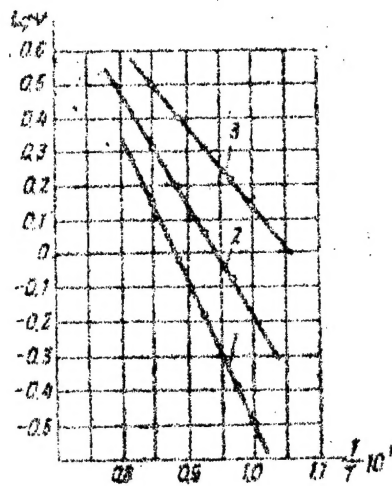


Figure 8. The dependence of the velocity ( $V$ ) of the reaction of the reduction of titanium tetrachloride with magnesium on the temperature.

- 1 - delivery of  $\text{TiCl}_4$  - 2  $\text{cm}^3$ ;
- 2 - delivery of  $\text{TiCl}_4$  - 5  $\text{cm}^3$ ;
- 3 - delivery of  $\text{TiCl}_4$  - 10  $\text{cm}^3$ .

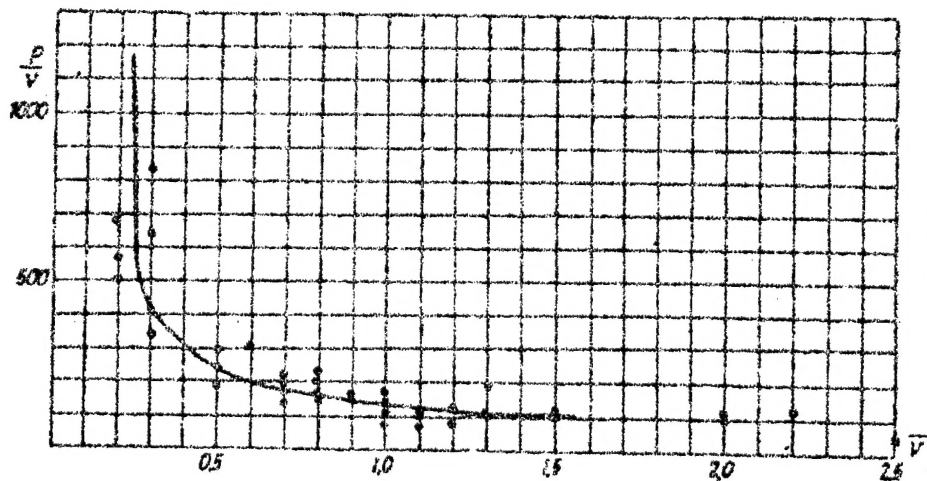


Figure 9. The relationship between the specific pressure ( $p$ ) and the rate of delivery of  $\text{TiCl}_4$  ( $v$ ) in small-capacity reduction apparatuses.

Pressure, millimeters  
of mercury  $1.5 \times 10^{-1}$

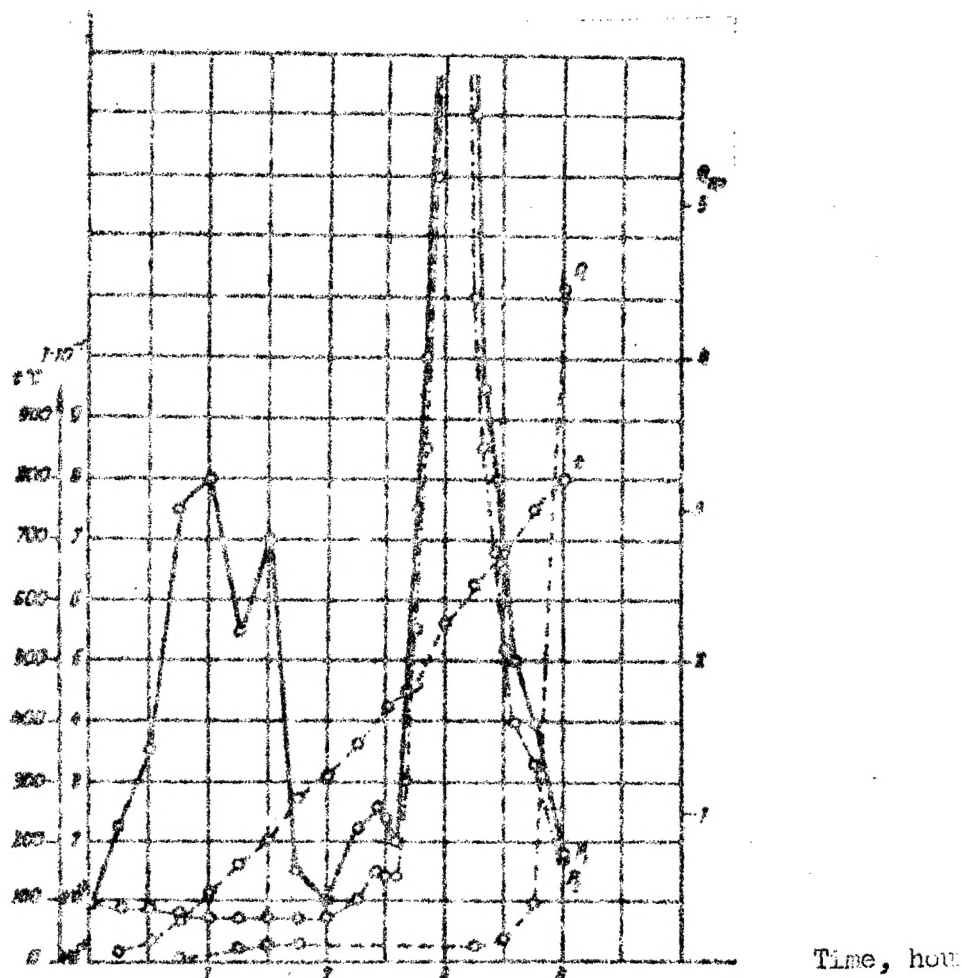


Figure 10. A study of the processes going on in the separation of the reaction mass, obtained by reducing titanium tetrachloride with magnesium:  
 Q - change in weight;  
 t - change in temperature;  
 P<sub>1</sub> - total pressure of water vapor and hydrogen in the system;  
 P<sub>2</sub> - change in the partial pressure of hydrogen in the system.